



US009290828B2

(12) **United States Patent**
Boudreault et al.

(10) **Patent No.:** **US 9,290,828 B2**
(45) **Date of Patent:** **Mar. 22, 2016**

(54) **PROCESSES FOR PREPARING TITANIUM OXIDE AND VARIOUS OTHER PRODUCTS**

(52) **U.S. Cl.**
CPC **C22B 21/0015** (2013.01); **C01B 7/035** (2013.01); **C01B 33/193** (2013.01);
(Continued)

(71) Applicant: **ORBITE ALUMINAE INC.**,
St-Laurent (CA)

(58) **Field of Classification Search**
None
See application file for complete search history.

(72) Inventors: **Richard Boudreault**, St-Laurent (CA);
Joël Fournier, Carignan (CA); **Denis Primeau**, Ste-Julie (CA);
Marie-Maxime Labrecque-Gilbert,
Laval (CA)

(56) **References Cited**

(73) Assignee: **ORBITE TECHNOLOGIES INC.**,
St-Laurent (CA)

U.S. PATENT DOCUMENTS

558,726 A 4/1896 Gooch
650,763 A 5/1900 Raynaud

(*) Notice: Subject to any disclaimer, the term of this patent is extended or adjusted under 35 U.S.C. 154(b) by 0 days.

(Continued)

FOREIGN PATENT DOCUMENTS

(21) Appl. No.: **14/413,940**

AU 631226 2/1991
AU 4375001 12/2001
CA 1066872 11/1979
CA 1088961 11/1980

(22) PCT Filed: **Jul. 12, 2013**

(Continued)

(86) PCT No.: **PCT/CA2013/000638**

OTHER PUBLICATIONS

§ 371 (c)(1),

(2) Date: **Jan. 9, 2015**

An English translation of Zhang et al., "Research on the Kinetics of Alumina from Kaolinite Leaching in Sulphuric Acid", Journal of Hefei University of Technology, vol. 24, No. 1, Feb. 2001, pp. 71-74.

(87) PCT Pub. No.: **WO2014/008586**

PCT Pub. Date: **Jan. 16, 2014**

(Continued)

(65) **Prior Publication Data**

US 2015/0159239 A1 Jun. 11, 2015

Primary Examiner — Melissa Swain

(74) *Attorney, Agent, or Firm* — Bereskin & Parr LLP/S.E.N.C.R.L., s.r.l.

Related U.S. Application Data

(63) Continuation-in-part of application No. PCT/CA2012/000871, filed on Sep. 17, 2012, and a continuation-in-part of application No. PCT/CA2013/000021, filed on Jan. 10, 2013.

(60) Provisional application No. 61/670,645, filed on Jul. 12, 2012, provisional application No. 61/726,971, filed on Nov. 15, 2012.

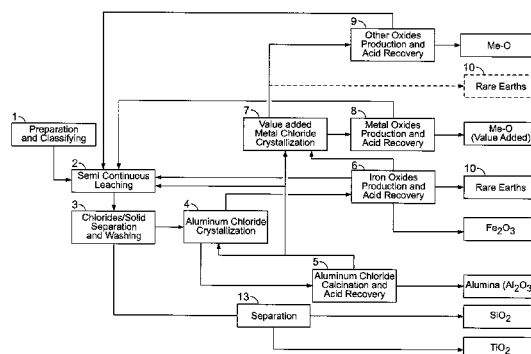
(51) **Int. Cl.**
C22B 21/00 (2006.01)
C01G 23/047 (2006.01)

(57) **ABSTRACT**

There are provided processes for preparing various products from various materials. For example, such processes are effective for extracting titanium and various other metals from various materials, thereby allowing for preparing products such as titanium chloride and titanium oxide. These processes can comprise leaching the starting material with HCl so as to obtain a leachate and a solid. The solid can be treated so as to substantially selectively extract titanium therefrom while the leachate can be treated so as to substantially selectively recover a first metal chloride therefrom.

(Continued)

22 Claims, 10 Drawing Sheets



(51)	Int. Cl.		3,658,483 A	4/1972	Lienau et al.
	C01F 7/56	(2006.01)	3,682,592 A	8/1972	Kovacs
	C22B 34/12	(2006.01)	3,751,553 A	8/1973	Oslo et al.
	C01F 7/22	(2006.01)	3,816,605 A	6/1974	Schwandorf
	C01F 7/30	(2006.01)	3,852,430 A	12/1974	Lienau et al.
	C01G 23/02	(2006.01)	3,862,293 A	1/1975	Maurel et al.
	C01G 49/06	(2006.01)	3,903,239 A	9/1975	Berkovich
	C01G 53/04	(2006.01)	3,922,164 A	11/1975	Reid et al.
	C01B 7/03	(2006.01)	3,944,648 A	3/1976	Solymar et al.
	C01B 33/193	(2006.01)	3,946,103 A	3/1976	Hund
	C22B 3/10	(2006.01)	3,966,909 A	6/1976	Grunig et al.
	C22B 3/44	(2006.01)	3,983,212 A	9/1976	Lowenstein et al.
	C22B 26/22	(2006.01)	4,042,664 A	8/1977	Cardwell et al.
	C22B 59/00	(2006.01)	4,045,537 A	8/1977	Hrishikesan
			4,048,285 A	9/1977	Szepesi et al.
			4,069,296 A	1/1978	Huang
			4,098,868 A	7/1978	Tolley
(52)	U.S. Cl.		4,107,281 A	8/1978	Reh et al.
	CPC	C01F 7/22 (2013.01); C01F 7/306	4,110,399 A	8/1978	Gaudernack et al.
		(2013.01); C01F 7/56 (2013.01); C01G 23/02	4,124,680 A	11/1978	Cohen et al.
		(2013.01); C01G 23/047 (2013.01); C01G	4,130,627 A	12/1978	Russ et al.
		49/06 (2013.01); C01G 53/04 (2013.01); C22B	4,133,677 A	1/1979	Matsui et al.
		3/10 (2013.01); C22B 3/44 (2013.01); C22B	4,158,042 A	6/1979	Deutschman
		26/22 (2013.01); C22B 34/1245 (2013.01);	4,172,879 A	10/1979	Miller et al.
		C22B 34/1259 (2013.01); C22B 59/00	4,177,242 A	12/1979	Cohen et al.
		(2013.01)	4,193,968 A	3/1980	Sullivan et al.
			4,198,231 A	4/1980	Gusset
			4,222,989 A *	9/1980	Belsky et al. 423/126
			4,224,287 A	9/1980	Ziegenbalg et al.
			4,226,844 A	10/1980	Reh et al.
(56)	References Cited		4,233,273 A	11/1980	Meyer et al.
	U.S. PATENT DOCUMENTS		4,237,102 A	12/1980	Cohen et al.
			4,239,735 A	12/1980	Eisele et al.
			4,241,030 A	12/1980	Cohen et al.
			4,259,311 A	3/1981	Shah
			4,297,326 A	10/1981	Gjelsvik et al.
			4,318,896 A	3/1982	Schoonover
			4,362,703 A	12/1982	Boybay et al.
			4,370,422 A	1/1983	Panda et al.
			4,378,275 A	3/1983	Adamson et al.
			4,392,987 A	7/1983	Laine et al.
			4,402,932 A	9/1983	Miller et al.
			4,414,196 A	11/1983	Matsumoto et al.
			4,435,365 A	3/1984	Morris
			4,437,994 A	3/1984	Baker
			4,465,566 A	8/1984	Loutfy et al.
			4,465,659 A	8/1984	Cambridge et al.
			4,486,393 A	12/1984	Baksa et al.
			4,490,338 A	12/1984	De Schepper et al.
			4,530,819 A	7/1985	Czeglédi et al.
			4,560,541 A	12/1985	Davis
			4,567,026 A	1/1986	Liosowyj
			4,585,645 A	4/1986	Sucech
			4,634,581 A	1/1987	Cambridge et al.
			4,652,433 A	3/1987	Ashworth et al.
			4,676,838 A	6/1987	Franz et al.
			4,710,369 A	12/1987	Bergman
			4,741,831 A	5/1988	Grinstead
			4,797,271 A	1/1989	Fleming et al.
			4,816,233 A	3/1989	Rourke et al.
			4,820,498 A	4/1989	Newkirk
			4,826,671 A	5/1989	Arndt et al.
			4,830,507 A	5/1989	Bagatto et al.
			4,898,719 A	2/1990	Rourke et al.
			4,913,884 A	4/1990	Feuling
			4,938,871 A	7/1990	Musikas et al.
			4,965,053 A	10/1990	Herchenroeder et al.
			4,968,504 A	11/1990	Rourke et al.
			4,980,141 A	12/1990	Kimura et al.
			4,988,487 A	1/1991	Lai et al.
			4,995,984 A	2/1991	Barkatt et al.
			5,006,753 A	4/1991	Hasker et al.
			5,008,089 A	4/1991	Moody et al.
			5,011,665 A	4/1991	Cailly et al.
			5,015,447 A	5/1991	Fulford et al.
			5,019,362 A	5/1991	Rourke et al.
			5,030,424 A	7/1991	Fulford et al.
			5,035,365 A	7/1991	Birmingham
			5,037,608 A	8/1991	Tarcy et al.

(56)

References Cited**U.S. PATENT DOCUMENTS**

5,039,336 A 8/1991 Feuling
 5,043,077 A 8/1991 Chandler et al.
 5,045,209 A 9/1991 Snyder et al.
 5,053,144 A 10/1991 Szirmai et al.
 5,061,474 A 10/1991 Pauli et al.
 5,071,472 A 12/1991 Traut et al.
 5,080,803 A 1/1992 Bagatto et al.
 5,091,159 A 2/1992 Connelly et al.
 5,093,091 A 3/1992 Dauplaise et al.
 5,104,544 A 4/1992 Shimizu et al.
 5,106,797 A 4/1992 Allaire
 5,112,534 A 5/1992 Guon et al.
 5,120,513 A 6/1992 Moody et al.
 5,124,008 A 6/1992 Rendall et al.
 5,149,412 A 9/1992 Allaire
 5,160,482 A 11/1992 Ash et al.
 5,180,563 A 1/1993 Lai et al.
 5,188,809 A 2/1993 Crocker et al.
 5,192,443 A 3/1993 Delloye et al.
 5,244,649 A 9/1993 Ostertag et al.
 5,274,129 A 12/1993 Natale et al.
 5,368,736 A 11/1994 Horwitz et al.
 5,409,677 A 4/1995 Zinn
 5,409,678 A 4/1995 Smith et al.
 5,433,931 A 7/1995 Bosserman
 5,443,618 A 8/1995 Chapman
 5,492,680 A 2/1996 Odekirk
 5,500,043 A 3/1996 Harada et al.
 5,505,857 A 4/1996 Misra et al.
 5,512,256 A 4/1996 Bray et al.
 5,531,970 A 7/1996 Carlson
 5,585,080 A 12/1996 Andersen et al.
 5,597,529 A 1/1997 Tack
 5,622,679 A 4/1997 Yuan et al.
 5,632,963 A 5/1997 Schwab et al.
 5,639,433 A 6/1997 Yuan et al.
 5,645,652 A 7/1997 Okinaka et al.
 5,665,244 A 9/1997 Rothenberg et al.
 5,720,882 A 2/1998 Stendahl et al.
 5,723,097 A 3/1998 Barnett et al.
 5,766,478 A 6/1998 Smith et al.
 5,787,332 A 7/1998 Black et al.
 5,792,330 A 8/1998 Petersen et al.
 5,795,482 A 8/1998 Ehle et al.
 5,876,584 A 3/1999 Cortellini
 5,885,545 A 3/1999 Pitzer
 5,904,856 A 5/1999 Kvant et al.
 5,911,967 A 6/1999 Ruthner
 5,922,403 A 7/1999 Tecle
 5,942,199 A 8/1999 Jokinen et al.
 5,955,042 A 9/1999 Barnett et al.
 5,962,125 A 10/1999 Masaki
 5,993,758 A * 11/1999 Nehari et al. 423/126
 5,997,828 A 12/1999 Rendall
 6,033,579 A 3/2000 Riemer et al.
 6,045,631 A 4/2000 Tarcy et al.
 6,077,486 A 6/2000 Spitzer
 6,093,376 A 7/2000 Moore
 6,153,157 A 11/2000 McLaughlin
 6,214,306 B1 4/2001 Aubert et al.
 6,221,233 B1 4/2001 Rendall
 6,238,566 B1 5/2001 Yoshida et al.
 6,248,302 B1 6/2001 Barnett et al.
 6,254,782 B1 7/2001 Kreisler
 6,267,936 B1 7/2001 Delmas et al.
 6,302,952 B1 10/2001 Mobbs et al.
 6,309,441 B1 10/2001 Benz et al.
 6,312,653 B1 11/2001 Delmau et al.
 6,337,061 B1 1/2002 Iyatomi et al.
 6,348,154 B1 2/2002 Stewart
 6,383,255 B1 5/2002 Sundkvist
 6,395,062 B2 5/2002 Olafson et al.
 6,395,242 B1 5/2002 Allen et al.
 6,406,676 B1 6/2002 Sundkvist
 6,447,738 B1 9/2002 Rendall et al.

6,468,483 B2 10/2002 Barnett et al.
 6,500,396 B1 12/2002 Lakshmanan et al.
 6,524,549 B1 2/2003 Mohri et al.
 6,565,733 B1 5/2003 Sportel et al.
 6,576,204 B2 6/2003 Johansen
 6,716,353 B1 4/2004 Mirzadeh et al.
 6,843,970 B1 1/2005 Hard
 6,893,474 B2 5/2005 Jäfverström et al.
 7,090,809 B2 8/2006 Harel et al.
 7,118,719 B2 10/2006 Fugleberg
 7,182,931 B2 2/2007 Turnbaugh, Jr. et al.
 7,220,394 B2 5/2007 Sreeram et al.
 7,282,187 B1 10/2007 Brown et al.
 7,294,319 B2 11/2007 Lahtinen et al.
 7,381,690 B1 6/2008 Ding et al.
 7,442,361 B1 10/2008 Gloeckler et al.
 7,498,005 B2 3/2009 Yadav
 7,651,676 B2 1/2010 Beaulieu et al.
 7,781,365 B2 8/2010 Okamoto
 7,837,961 B2 11/2010 Boudreault et al.
 7,892,426 B2 2/2011 Hayashi et al.
 7,906,097 B2 3/2011 Beaulieu et al.
 8,038,969 B2 10/2011 Kondo et al.
 8,147,795 B2 4/2012 Dolling et al.
 8,216,532 B1 7/2012 Vierheilg
 8,241,594 B2 8/2012 Boudreault et al.
 8,287,826 B2 10/2012 Pettey
 8,337,789 B2 12/2012 Boudreault et al.
 8,568,671 B2 10/2013 Guo et al.
 2002/0050230 A1 5/2002 Meisen
 2002/0071802 A1 6/2002 Fulton et al.
 2003/0075021 A1 4/2003 Young et al.
 2003/0152502 A1 8/2003 Lewis et al.
 2003/0183043 A1 10/2003 Wai et al.
 2004/0042945 A1 3/2004 Rao et al.
 2004/0062695 A1 4/2004 Horwitz et al.
 2005/0166706 A1 8/2005 Withers et al.
 2006/0018813 A1 1/2006 Bray
 2006/0066998 A1 3/2006 Ishiguro
 2007/0062669 A1 3/2007 Song et al.
 2007/0278106 A1 12/2007 Shaw
 2008/0047395 A1 2/2008 Liu et al.
 2008/0069748 A1 3/2008 Lien et al.
 2008/0115627 A1 5/2008 Wang et al.
 2008/0286182 A1 11/2008 Costa et al.
 2009/0241731 A1 10/2009 Pereira et al.
 2009/0272230 A1 11/2009 Mackowski et al.
 2010/0018347 A1 1/2010 Holden et al.
 2010/0078382 A1 4/2010 Naganawa et al.
 2010/0129277 A1 5/2010 Kondo et al.
 2010/0150799 A1 6/2010 Boudreault et al.
 2010/0160144 A1 6/2010 Kim et al.
 2010/0260640 A1 10/2010 Shindo et al.
 2010/0278720 A1 11/2010 Wong et al.
 2010/0319491 A1 12/2010 Sugahara et al.
 2010/0329970 A1 12/2010 Lian et al.
 2011/0017020 A1 1/2011 Homma et al.
 2011/0044869 A1 2/2011 Boudreault et al.
 2011/0120267 A1 5/2011 Roche
 2011/0182786 A1 7/2011 Burba, III
 2012/0073407 A1 3/2012 Drinkard, Jr. et al.
 2012/0237418 A1 9/2012 Boudreault et al.
 2013/0052103 A1 2/2013 Boudreault et al.
 2014/0341790 A1 11/2014 Kasaini
 2014/0356262 A1 12/2014 Ruth et al.

FOREIGN PATENT DOCUMENTS

CA 1136380 11/1982
 CA 1176470 10/1984
 CA 1224327 7/1987
 CA 1226719 9/1987
 CA 2027519 4/1991
 CA 2027973 4/1991
 CA 2029623 5/1991
 CA 2036058 8/1991
 CA 2097809 7/1992
 CA 2137249 12/1993
 CA 2122364 2/1994

(56)

References Cited

FOREIGN PATENT DOCUMENTS

CA	2156295	9/1994	EP	1496063	1/2005
CA	2160488	11/1994	EP	2241649	10/2010
CA	2193726	1/1996	EP	2298944	3/2011
CA	2159534	4/1996	FR	2319579	2/1977
CA	2167890	7/1996	FR	2600635	12/1987
CA	2240067	6/1997	GB	153500	11/1920
CA	2251433	4/1999	GB	159086	2/1921
CA	2360447	8/2000	GB	195295	3/1923
CA	2306015	12/2000	GB	230916	3/1925
CA	2309225	12/2000	GB	240834	5/1926
CA	2377600	1/2001	GB	241184	5/1926
CA	2317692	3/2001	GB	273999	7/1927
CA	2391394	5/2001	GB	409710	5/1934
CA	2400673	8/2001	GB	470305	8/1937
CA	2429889	6/2002	GB	480921	3/1938
CA	2431466	6/2002	GB	490099	8/1938
CA	2433448	7/2002	GB	574818	1/1946
CA	2189631	11/2002	GB	745601	2/1956
CA	2454812	2/2003	GB	798750	7/1958
CA	2468885	7/2003	GB	857245	12/1960
CA	2471179	7/2003	GB	858026	1/1961
CA	2484134	11/2003	GB	1021326	3/1966
CA	2467288	11/2004	GB	1056488	1/1967
CA	2548225	11/2004	GB	1307319	2/1973
CA	2385775	5/2005	GB	2013164	8/1979
CA	2556613	8/2005	GB	1552918	9/1979
CA	2572190	1/2006	GB	2018230	10/1979
CA	2597440	8/2006	GB	2238813	6/1991
CA	2521817	3/2007	JP	04354836	A * 12/1992
CA	2624612	4/2007	JP	05287405	11/1993
CA	2629167	5/2007	JP	6056429	3/1994
CA	2639796	6/2007	OA	010034	10/1996
CA	2636379	7/2007	WO	8603521	6/1986
CA	2641919	8/2007	WO	9103424	3/1991
CA	2538962	9/2007	WO	9213637	8/1992
CA	2608973	1/2008	WO	9313017	7/1993
CA	2610918	2/2008	WO	9418122	8/1994
CA	2659449	2/2008	WO	9600698	1/1996
CA	2684696	11/2008	WO	9621619	7/1996
CA	2685369	11/2008	WO	9624555	8/1996
CA	2711013	11/2008	WO	WO 9624555	A1 * 8/1996
CA	2697789	3/2009	WO	9722554	6/1997
CA	2725391	11/2009	WO	0104366	1/2001
CA	2678724	3/2010	WO	2004056468	7/2004
CA	2745572	7/2010	WO	2004056471	7/2004
CA	2747370	7/2010	WO	2004101833	11/2004
CA	2667029	11/2010	WO	2005123591	12/2005
CA	2667033	11/2010	WO	2006084682	8/2006
CA	2678276	3/2011	WO	2007074207	7/2007
CA	2773571	3/2011	WO	2007079532	7/2007
CA	2788965	8/2011	WO	2008067594	6/2008
CA	2797561	11/2011	WO	2008104250	9/2008
CA	2834356	12/2012	WO	2008141423	11/2008
CA	2884787	4/2013	WO	2008154995	12/2008
CA	2860491	8/2013	WO	2009085514	7/2009
CN	1099424	3/1995	WO	2009153321	12/2009
CN	1923730	3/2007	WO	2010002059	1/2010
CN	101773925	7/2010	WO	2010009512	1/2010
CN	101792185	8/2010	WO	2010056742	5/2010
DE	19903011	8/2000	WO	2010079369	7/2010
EP	157503	10/1985	WO	2010133284	11/2010
EP	0054976	7/1986	WO	2011094858	8/2011
EP	0238185	9/1987	WO	2011100820	8/2011
EP	0279672	8/1988	WO	2011100821	8/2011
EP	0327234	8/1989	WO	2011147867	12/2011
EP	0382383	8/1990	WO	2014029031	2/2012
EP	0399786	11/1990	WO	2012126092	9/2012
EP	508676	10/1992	WO	2012145797	11/2012
EP	466338	12/1995	WO	2012149642	11/2012
EP	0449942	4/1996	WO	2013037054	3/2013
EP	0775753	5/1997	WO	2013142957	10/2013
EP	0829454	3/1998	WO	2014094155	6/2014
EP	0692035	4/1998	WO	2014094157	6/2014
EP	0834584	4/1998	WO	2014124539	8/2014
EP	999185	5/2000			

OTHER PUBLICATIONS

An English translation of Zhang et al., "Research of the Controlling Steps of the Reaction of Kaolin and Hydrochloric Acid", Journal of Hefei University of Technology, vol. 21, No. 1, Feb. 1998, pp. 50-53.

(56)

References Cited**OTHER PUBLICATIONS**

An English translation of Zhang et al., "Kinetics Research on Alumina in Kaolinite Leached by Hydrochloric Acid", *Journal of Hefei University of Technology*, vol. 22, No. 2, Apr. 1999, pp. 33-36.

Certification of translation from Park IP Translations dated May 14, 2012.

An English Abstract of CN101450811 "Method for extracting alumina from coal gangue", published on Jun. 10, 2009.

An English Abstract of JP2001162108 "Method for Manufacturing Iron-Aluminum Combined Flocculant", published on Jun. 19, 2001.

English Translation of Abstract of CN101462757, "Preparation of nano Na-beat-alumina powder", Jun. 24, 2009.

An English Abstract of JP57145027 "Preparation of Granular Alumina", published on Sep. 7, 1982.

An English Abstract of CA1065068 "Method of Selectively Precipitating Metals From Solutions", published on Oct. 23, 1979.

English Translation of Abstract of CN101289705, "Process for abstracting vanadium from iron-smeltin waste slag of vanadium-containing iron ore", Jul. 14, 2010.

English Translation of Abstract of CN102220487, "Method for extracting vanadium and aluminum from vanadium-containing stone coal and clay vanadium ore", Oct. 19, 2011.

English Translation of Abstract of CN102241410, "Ecological and Comprehensive Utilization Method of Coal Ash", Nov. 16, 2011.

English Translation of Abstract of RU2363748, "Method of Producing Aluminium", Aug. 10, 2009.

English Translation of Abstract of ES2194586, "Separation procedure for contaminatory metals present in acid solutions involves liquid-liquid extraction with mixtures based on phosphonated dialkyl alkyl, trialkyl phosphates and acid phosphates", Mar. 1, 2005.

English Translation of Abstract of WO2009005115, "Composition for promotion of reduction in size of adipocyte", Jan. 8, 2009.

English Translation of Abstract of CN101434484, "Processes for producing alumina ceramic valve body and use thereof", May 20, 2009.

English Translation of Abstract of CN101045543, "Method for preparing sheet alumina using coal series kaolin rock or flyash as raw material", Oct. 3, 2007.

English Translation of Abstract of CN101249965, "Method for preparing ultra-fine white carbon black and nano alumina by using kaolinite as raw material", Aug. 27, 2008.

English Translation of Abstract of CN101045538, "Method for preparing modified silicon oxide using coal series kaolin rock or flyash", Oct. 3, 2007.

Teeisele, "Primary Metal Production", Dec. 3, 2007.

US EPA, "Alumina & Aluminum", Office of Resource Conservation and Recovery, Apr. 2, 2012.

Abstract of Dash et al., "Acid dissolution of alumina from waste aluminium dross", *Hydrometallurgy*, vol. 92, issues 1-2, May 2008, pp. 48-53.

Copson et al., "Extraction of Alumina from Clays by the Lime-sinter Modification of the Pedersen Process", New York Meeting, Feb. 1944, pp. 241-254.

Wei, X. et al., "Recovery of Iron and Aluminium from Acid Mine Drainage by selective precipitation", *Environmental Engineering Science*, vol. 22, No. 6, 2005, pp. 745-755.

Ajemba et al., "Application of the Shrinking Core Model to the Analysis of Alumina Leaching From UKPOR Clay Using Nitric Acid", *International Journal of Engineering Research & Technology (IJERT)*, ISSN: 2278-0181, vol. 1 Issue 3, May 2012.

Zhou et al., "Extraction of Scandium from red mud by modified activated carbon and kinetics study", *Rare Metals*, vol. 27, No. 3, Jun. 2008, pp. 223-227.

Ouellet, Dissertation 9689, (Extraction de l'alumine de l'argile de la région de Murdochville, Québec, Canada), "Extraction of Alumina from Clay in the Murdochville region of Quebec", Canada—Universite Laval—Original French Version, Oct. 2004.

Ouellet, Dissertation 9689, (Extraction de l'alumine de l'argile de la région de Murdochville, Québec, Canada), "Extraction of Alumina

from Clay in the Murdochville region of Quebec", Canada—Universite Laval—English Translation, Oct. 2004.

Translator Certification—Sep. 10, 2012.

English Abstract BE1019347, "Hydrometallurgical Reactor", published on Jun. 5, 2012.

Abstract Kao et al., "Solvent extraction of La(III) and Nd(III) from nitrate solutions with 2-ethylhexylphosphonic acid mono-2-ethylhexyl ester", *Chemical Engineering Journal*, vol. 119, Issues 2-3, Jun. 15, 2006, pp. 167-174.

English Abstract CN102690954, "Back extraction and removement method for aluminium", published on Sep. 26, 2012.

English Abstract CN101781719, "Method for recovering rare earth from oil shale waste slag", published on Jul. 21, 2010.

English Abstract CN102643985, "Method for extracting valuable metals from high-iron bauxite with step-by-step acid leaching", published on Aug. 22, 2012.

English Abstract CN102628105, "Method for comprehensively recycling and using baric waste slag in refined aluminum production process", published on Aug. 8, 2012.

English Abstract of WO 2007122720, published on Nov. 1, 2007.

English Abstract of WO 2004085719, published on Oct. 7, 2004.

English Abstract of SU 1 734 395, published on Oct. 27, 1996.

English Abstract of RU2416655, published on Apr. 20, 2011.

English Abstract of RU2008113385, published on Oct. 20, 2009.

English Abstract of RU2361941, published on Jul. 20, 2009.

English Abstract of RU2257348, published on Jul. 27, 2005.

English Abstract of RU2247788, published on Mar. 10, 2005.

English Abstract of RU2236375, published on Sep. 20, 2004.

English Abstract of RU2205242, published on May 27, 2003.

English Abstract of RU2201988, published on Apr. 10, 2003.

English Abstract of RU2196184, published on Jan. 10, 2003.

English Abstract of RU2189358, published on Sep. 20, 2002.

English Abstract of RU2176680, published on Dec. 10, 2001.

English Abstract of RU2162898, published on Feb. 10, 2001.

English Abstract of RU2162112, published on Jan. 20, 2001.

English Abstract of RU2158170, published on Oct. 27, 2000.

English Abstract of RU2147623, published on Apr. 20, 2000.

English Abstract of RU2147622, published on Apr. 20, 2000.

English Abstract of RU2140998, published on Nov. 10, 1999.

English Abstract of RU2119816, published on Oct. 10, 1998.

English Abstract of KR20070028987, published on Mar. 13, 2007.

English Abstract of JP9324227, published on Dec. 16, 1997.

English Abstract of JP9324192, published on Dec. 16, 1997.

English Abstract of JP9291320, published on Nov. 11, 1997.

English Abstract of JP9249672, published on Sep. 22, 1997.

English Abstract of JP9248463, published on Sep. 22, 1997.

English Abstract of JP9208222, published on Aug. 12, 1997.

English Abstract of JP9194211, published on Jul. 29, 1997.

English Abstract of JP9176756, published on Jul. 8, 1997.

English Abstract of JP9143589, published on Jun. 3, 1997.

English Abstract of JP8232026, published on Sep. 10, 1996.

English Abstract of JP5051208, published on Mar. 2, 1993.

English Abstract of JP4198017, published on Jul. 17, 1992.

English Abstract of JP4183832, published on Jun. 30, 1992.

English Abstract of JP4046660, published on Feb. 17, 1992.

English Abstract of JP3173725, published on Jul. 29, 1991.

English Abstract of JP2179835, published on Jul. 12, 1990.

English Abstract of JP2080530, published on Mar. 20, 1990.

English Abstract of JP2011116622, published on Jun. 16, 2011.

English Abstract of JP2011046588, published on Mar. 10, 2011.

English Abstract of JP2010270359, published on Dec. 2, 2010.

English Abstract of JP2008194684, published on Aug. 28, 2008.

English Abstract of JP2007327126, published on Dec. 20, 2007.

English Abstract of JP2007254822, published on Oct. 4, 2007.

English Abstract of JP2006348359, published on Dec. 28, 2006.

English Abstract of JP2006028187, published on Feb. 2, 2006.

English Abstract of JP2005139047, published on Jun. 2, 2005.

English Abstract of JP2000313928, published on Nov. 14, 2000.

English Abstract of JP10158629, published on Jun. 16, 1998.

English Abstract of JP10121164, published on May 12, 1998.

English Abstract of EP1817437, published on Aug. 15, 2007.

English Abstract of CN2292806, published on Sep. 30, 1998.

English Abstract of CN1986895, published on Jun. 27, 2007.

(56)

References Cited

OTHER PUBLICATIONS

- English Abstract of CN1796608, published on Jul. 5, 2006.
 English Abstract of CN1699609, published on Nov. 23, 2005.
 English Abstract of CN1410599, published on Apr. 16, 2003.
 English Abstract of CN1397653, published on Feb. 19, 2003.
 English Abstract of CN1192479, published on Sep. 9, 1998.
 English Abstract of CN1478600, published on Mar. 3, 2004.
 English Abstract of CN1140148, published on Jan. 15, 1997.
 English Abstract of CN1127791, published on Jul. 31, 1996.
 English Abstract of CN1131200, published on Sep. 18, 1996.
 English Abstract of CN1061246, published on May 20, 1992.
 English Abstract of CN1043752, published on Jul. 11, 1990.
 English Abstract of CN102153128, published on Aug. 17, 2011.
 English Abstract of CN102139943, published on Aug. 3, 2011.
 English Abstract of CN102127641, published on Jul. 20, 2011.
 English Abstract of CN101407879, published on May 25, 2011.
 English Abstract of CN102071315, published on May 25, 2011.
 English Abstract of CN102061392, published on May 18, 2011.
 English Abstract of CN102030355, published on Apr. 27, 2011.
 English Abstract of CN102021343, published on Apr. 20, 2011.
 English Abstract of CN102011010, published on Apr. 13, 2011.
 English Abstract of CN101824555, published on Sep. 8, 2010.
 English Abstract of CN1045812, published on Oct. 3, 1990.
 English Abstract of CN101407879, published on Apr. 15, 2009.
 English Abstract of CN101307384, published on Nov. 19, 2008.
 English Abstract of CN101161834, published on Apr. 16, 2008.
 English Abstract of CN1844421, published on Oct. 11, 2006.
 English Abstract of CN101182601, published on May 21, 2008.
 English Abstract of CN1043752C, published on Jun. 23, 1999.
 English Abstract of CN87101034, published on Aug. 28, 1991.
 English Abstract of RU2079431, published on May 20, 1997.
 English Abstract of RU2063458, published on Jul. 10, 1996.
 English Abstract of RU 2 048 565, published on Nov. 20, 1995.
 English Abstract of RU 2 040 587, published on Jul. 25, 1995.
 English Abstract of RU 2 034 074, published on Apr. 30, 1995.
 English Abstract of RU 2 031 168, published on Mar. 20, 1995.
 English Abstract of RU 2 020 175, published on Sep. 30, 1994.
 English Abstract of RU 2 010 876, published on Apr. 15, 1994.
 English Abstract of RU 2 094 374, published on Oct. 27, 1997.
 English Abstract of RU 2 081 831, published on Jun. 20, 1997.
 English Abstract of RU 2 070 596, published on Dec. 20, 1996.
 English Abstract of AU2008286599A1, "A process of smelting monazite rare earth ore rich in Fe", published on Feb. 19, 2009.
 English Abstract of KR820001546, "Production of Titanium Metal Valves", published on Aug. 31, 1982.
 English Abstract of KR100927466, published on Nov. 19, 2009.
 English Abstract of RU 2 069 180, published on Nov. 20, 1996.
 English Abstract of RU 2 068 392, published on Oct. 27, 1996.
 English Abstract of RU 2 062 810, published on Jun. 27, 1996.
 English Abstract of RU 2 055 828, published on Mar. 10, 1996.
 English Abstract of RU 2 049 728, published on Dec. 10, 1995.
 English Abstract of RU 1 704 483, published on Oct. 27, 1996.
 English Abstract of SU1567518, published on May 30, 1990.
 English Abstract of SU1424174, published on Jul. 23, 1991.
 English Abstract of RU2038309, published on Jun. 27, 1995.
 English Abstract of HU51574, published on May 28, 1990.
 English Abstract of JP2008253142, published on Oct. 23, 2008.
 English Abstract of JP2005152756, published on Jun. 16, 2005.
 English Abstract of JP2005082462, published on Mar. 31, 2005.
 English Abstract of JP2005219938, published on Aug. 18, 2005.
 English Abstract of JP2004036003, published on Feb. 5, 2004.
 English Abstract of WO2011092292, published on Aug. 4, 2011.
 English Abstract of CN101157453, published on Apr. 9, 2008.
 English Abstract of AU2737892, published on May 13, 1993.
 English Abstract of JPH09249420, published on Sep. 22, 1997.
 English Abstract of CN102515590, published on Jun. 27, 2012.
 English Abstract of CN102502745, published on Jun. 20, 2012.
 English Abstract of WO2007082447, published on Jul. 26, 2007.
 English Abstract of CN102694218, published on Sep. 26, 2012.
 English Abstract of RU2183225, published on Jun. 10, 2002.
 English Abstract of JP2009249674, published on Oct. 29, 2009.
 English Abstract of CN102719674, published on Oct. 10, 2012.
 English Abstract of 102680423, published on Sep. 19, 2012.
 English Abstract of JP2005112636, published on Apr. 28, 2005.
 English Abstract of CN101285127, published on Oct. 15, 2008.
 Australie Minerals & Mining Group LTD, "AMMG Updates Process Design for HPA Chemical Project", Oct. 3, 2014, pp. 1-4.
 English Abstract of CN1044126(C), "Stretched polypropylene film", published on Jul. 14, 1999.
 English Translation of CN102452677, published on May 16, 2012.
 English Abstract of CN101781719, "Method for recovering rare earth from oil shale waste slag", published on Jul. 21, 2010.
 Yatsenko et al., "Red Mud Pulp Carbonization with Scandium Extraction during alumina Production", ISSN 0040-5795, Theoretical Foundations of Chemical Engineering, Jan. 2010, vol. 44, No. 4, pp. 563-568.
 Wang et al., "Recovery of scandium from synthetic red mud leach solutions by solvent extraction with D2EHPA", Separation and Purification Technology (Feb. 2013), pp. 1-14.
 Andritz Metals, "Regeneration Systems for Hydrochloric Waste Pickling Solutions", Aug. 2011.
 Aluminium for Future Generations, "Bauxite Residual Management", <http://bauxite.world-aluminium.org/refining/bauxite-residue-management.html>, Jan. 2012.
 Vedanta Aluminium Limited, Lanjigarh, "Red Mud Filtration and Recovery of Valuable Metals", Aug. 2011, <http://marvels.engineeringwatch.in/eea025/>.
 Adham et al., "Fluid Bed Dehydration of Magnesium Chloride", Magnesium Technology 2012, TMS (The Minerals, Metals & Materials Society), Jan. 2012, pp. 49-53.
 Dutrizac et al., "Fundamentals of Serpentine Leaching in Hydrochloric Acid Media", Magnesium Technology 2000, The Minerals, Metals & Materials Society, Jan. 2000, pp. 41-51.
 Kumar et al., "Refining of a low-grade molybdenite concentrate", Hydrometallurgy 86 (Jan. 2007) 56-62.
 Wang et al., "Metallurgical processes for scandium recovery from various resources: A review", Hydrometallurgy 108 (Mar. 2011) 100-108.
 Hudson et al., "Direct Calcination of AlCl₃·6H₂O with Off-Gas use for Crystallization", United States Department of the Interior Bureau of Mines, Contract No. J0188096, Jan. 1979, pp. 1-28.
 Elsner et al., "Alumina via hydrochloric acid leaching of high silica bauxites—Process Development", Jan. 1984, pp. 411-429.
 Maysilles et al., "Aluminum Chloride Hexahydrate Crystallization by HCl Gas Sparging", U.S. Dept. of the Interior, Bureau of Mines, Jan. 1981, pp. 1-38.
 Brand et al., "Formation of α -Al₂O₃ by thermal decomposition of basic aluminum chlorides at low temperatures", Crystal Research and Technology Jan. 1989 (24) 671-675.
 Park et al., "Manufacture of low-soda alumina from clay", Industrial and Engineering Chemistry Jan. 1996 (35) 4379-4385.
 Yanagida et al., "The role of water vapor in formation of alpha alumina from transient alumina" Journal of Ceramic Association Japan Jan. 1966 (74) 371-77.
 Pijolat et al., "Influence of additives and water vapor on the transformation of transition aluminas into alpha alumina", Thermochimica Acta Dec. 1987 (122) 71-77.
 Bagwell et al., "Effect of seeding and water vapor on the nucleation and growth of α -Al₂O₃ from γ -Al₂O₃", Journal of the American Ceramic Society Jan. 1999 (82) 825-832.
 Hrabe et al., "The influence of water vapor on thermal transformations of boehmite", Journal of Material Research Jan. 1992 (7) 444-449.
 English Abstract of Petzold et al., "Thermoanalytical studies on the decomposition of aluminum chloride hexahydrate", Journal of thermal analysis Jan. 1981 (20) 71-86.
 Fang et al., "Recovery of gallium from coal fly ash", Hydrometallurgy 41 (Jan. 1996) 187-200.
 Gutiérrez et al., "Recovery of gallium from coal fly ash by a dual reactive extraction process", Waste Management & Research (Jan. 1997) 16, 371-382.

(56)

References Cited**OTHER PUBLICATIONS**

Gunnar et al., Abstract of "Extraction of iron compounds from wood from the Vasa", Department of Chemistry, Swedish University of Agricultural Sciences, vol. 60, No. 6, pp. 678-684, Jan. 2006.

Elmolla et al., "Effect of Photo-Fenton Operating Conditions on the Performance of Photo-Fenton-SBR Process for Recalcitrant Wastewater Treatment", *Journal of Applied Sciences* 10 (24): 3236-3242, Jun. 2010.

Cohen et al., "Precipitation of iron from concentrated chloride solutions: Literature observations, challenges and preliminary experimental results", *Minerals Engineering* 18 (Sep. 16, 2005), pp. 1344-1347.

Cablik, "Characterization and applications of red mud from bauxite processing", VSB—Technical University of Ostrava, Faculty of Mining and Geology, pp. 27-37, Jan. 2007.

Wahab et al., "Alumina Recovery From Iraqi Kaolinitic Clay by Hydrochloric Acid Route", *Iraqi Bulletin of Geology and Mining*, vol. 2, No. 1, Jan. 2006, pp. 67-76.

Khan et al., "Production of Aluminum Sulphate from Indigenous Bauxite without Precalcination of the Ore", *Jour. Chem. Soc. Pak.*, vol. 17, No. 4, pp. 213-216, Apr. 1995.

Sahoo et al., "Characterization of γ - and α -Fe₂O₃ nano powders synthesized by emulsion precipitation-calcination route and rheological behavior of α -Fe₂O₃", *International Journal of Engineering, Science and Technology*, vol. 2, No. 8, pp. 118-126, Jan. 2010.

Bharathi et al., "Highly mesoporous α -Fe₂O₃ nanostructures: preparation, characterization and improved photocatalytic performance towards Rhodamine B (RhB)", *J. Phys. D: Appl. Phys.* 43 015501, pp. 1-9, Jan. 2010.

"Industrial Inorganic Pigments", Wiley—VCH Verlag GmbH and Co. KGaA, pp. 105-112, Jan. 1993.

Andrieux et al., Abstract of: "Hydrothermal synthesis of dioctahedral smectites: The Al-Fe³⁺ chemical series: Part I: Influence of experimental conditions", Université de Poitiers, Nov. 2009.

Bazin et al., "Alumina from clays", Department of Mining, Metallurgical and Materials Engineering; Alcan International Limitee; Groupe Conseil PROCD Inc.; Conseil de Développement économique de Murdochville, pp. 24-38, Jan. 2005.

Aleksandrovich, "The receipt of alumina from clay materials", May 2011.

Al-Zahrani et al., "Extraction of Alumina from Local Clays by Hydrochloric Acid Process", *JKAU: Eng. Sci.*, vol. 20, No. 2, pp. 29-41, Jan. 2009.

Dutrizac et al., "The Precipitation of Hematite from Ferric Chloride Media at Atmospheric Pressure", *Mining and Mineral Sciences Laboratories*, vol. 30B, Dec. 12, 1999, pp. 993-1001.

Riveros et al., "The precipitation of hematite from ferric chloride media", *Mining and Mineral Sciences Laboratories, Hydrometallurgy* 46 (Jan. 1997), pp. 85-104.

Smirnov, V., "Alumina production in Russia Part I: Historical background", *Journal of Materials*, vol. 48, Issue 8, Jan. 1996, pp. 24-26.

Wang et al., "A novel recovery process of metal values from the cathode active materials of the lithium-ion secondary batteries", *Hydrometallurgy* 99 (Aug. 2009) 194-201.

Becze et al., "Precipitation of Hematite and Recovery of Hydrochloric Acid from Concentrated Chloride Solutions by a Novel Hydrolytic Decomposition Process", *TMS (The Minerals & Materials Society)*, Jan. 2001.

Demopoulos et al., "New Technologies for HCl Regeneration in Chloride Hydrometallurgy", *World of Metallurgy—ERZMETALL* 61 (Jan. 2008) No. 2.

English Abstract of CN103964480(A), "Process for Producing Aluminum Oxide by Using Hydrochloric Acid Method", published on Aug. 6, 2014.

English Abstract of CN203922759(U), "Device for Preparing Aluminum Oxide by Roasting Aluminum Chloride Hexahydrate Crystal through Rotary Kiln", published on Nov. 5, 2014.

English Abstract of EP0850881(A1), "Process and Apparatus for the Preparation of Iron Oxides from Solutions Containing Hydrochloric Acid Iron Oxide Chloride", published on Jul. 1, 1998.

English Abstract of JPH0543252A, "Method for Removing Halogen Radical in Ferric Oxide", published on Feb. 23, 1993.

English Abstract of JPH0656429(A), "Production of Plate-Like Iron Oxide Particulate Powder", published on Mar. 1, 1994.

English Abstract of WO2008070885(A2), "Method for Increasing the Specific Surface of Iron Oxides in Spray Roasting Plants", published on Jun. 19, 2008.

Englist Abstract of CN1099424A, "Method for Treating Blast Furnace Slag with Diluted Chlorhydric Acid", published on Mar. 1, 1995.

Weissenbaeck et al., "Development of Chloride Based Metal Extraction Techniques", Paper presented at ALTA May 28, 2013, Perth, WA.

Hoffman et al., "Development of a hydrochloric acid process for the production of alumina from clay", *Journal of research of the national bureau of standards* Dec. 1946 (37) 409-428.

Marchessaux et al., "Thermal decomposition of aluminum hexahydrate chloride (AlCl₃ × 6H₂O) for alumina production", *Light metals* Jan. 1979 (1) 189-204.

Miller et al., "Fluidized-bed decomposition of aluminum chloride hexahydrate", *Light Metals: Proceedings of Sessions, AIME Annual Meeting* (Warrendale, Pennsylvania), Jan. 1983 253-271.

Shanks et al., "Options in the HCl process for the production of alumina from clay", *Light Metals* Jan. 1986, R.E. Miller, Editor, p. 1089.

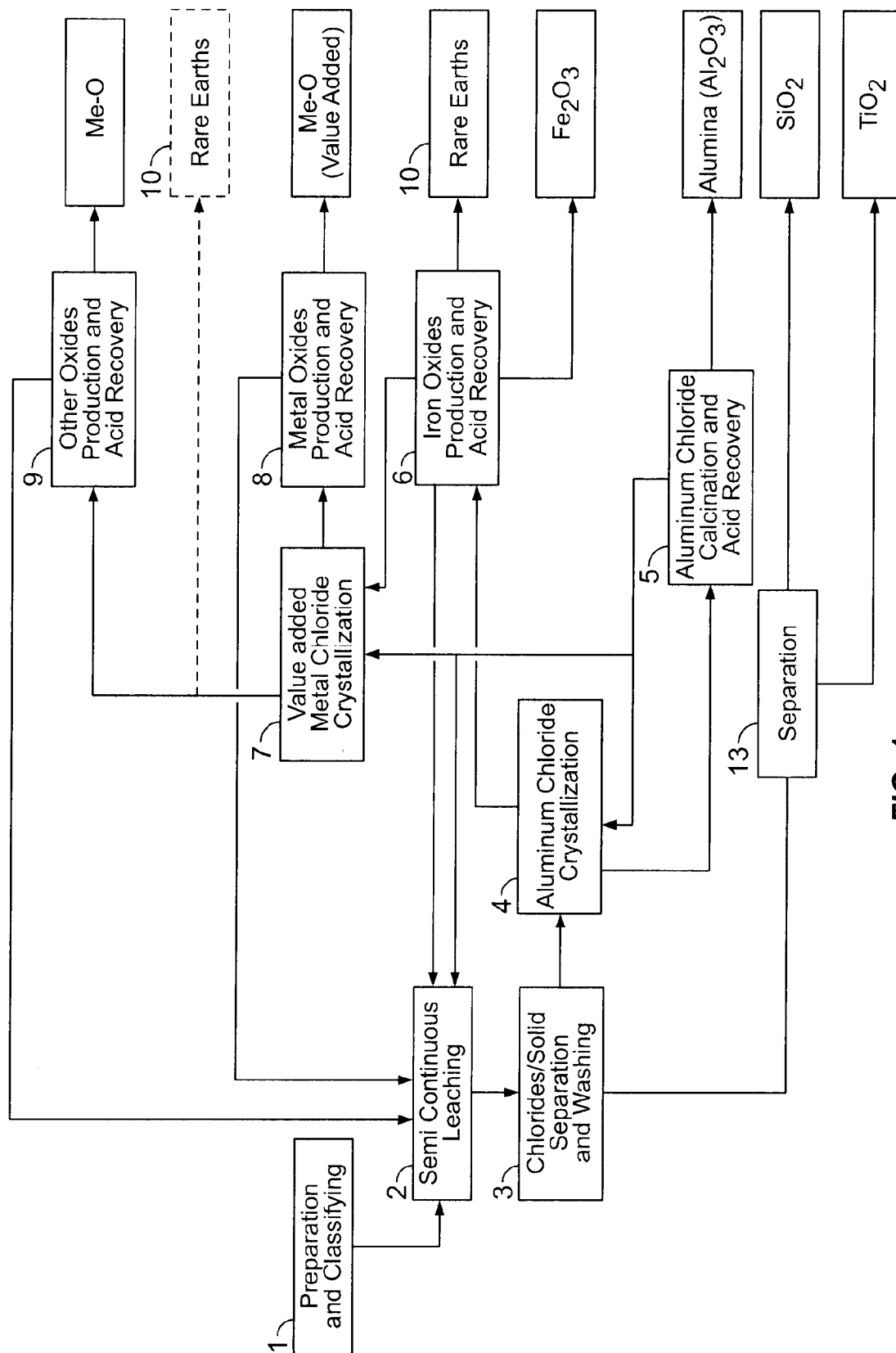
Bengston et al., "Alumina process feasibility study and preliminary pilot plant design. Task 3 report: preliminary design of 25 ton per day pilot plant.", vol. 1, *Process Technology and Costs*. Bureau of Mines Open File Report PB81-125031, Nov. 1979.

Liu et al., "High Purity Alumina Powders Extracted from Aluminum Dross by the Calcining-Leaching Process", *TMS (The Minerals, Metals & Materials Society)*, Jan. 2011, pp. 197-200.

Perander et al., "The Nature and Impacts of Fines in S melter-Grade Alumina." *Journal of the Minerals, Metals, & Materials Society* 61.11 (Nov. 2009).

Chi et al. Derwent Acc-No. 2010-L68306 for the patent family including CN 101811712A published Aug. 25, 2010. Abstract.

* cited by examiner



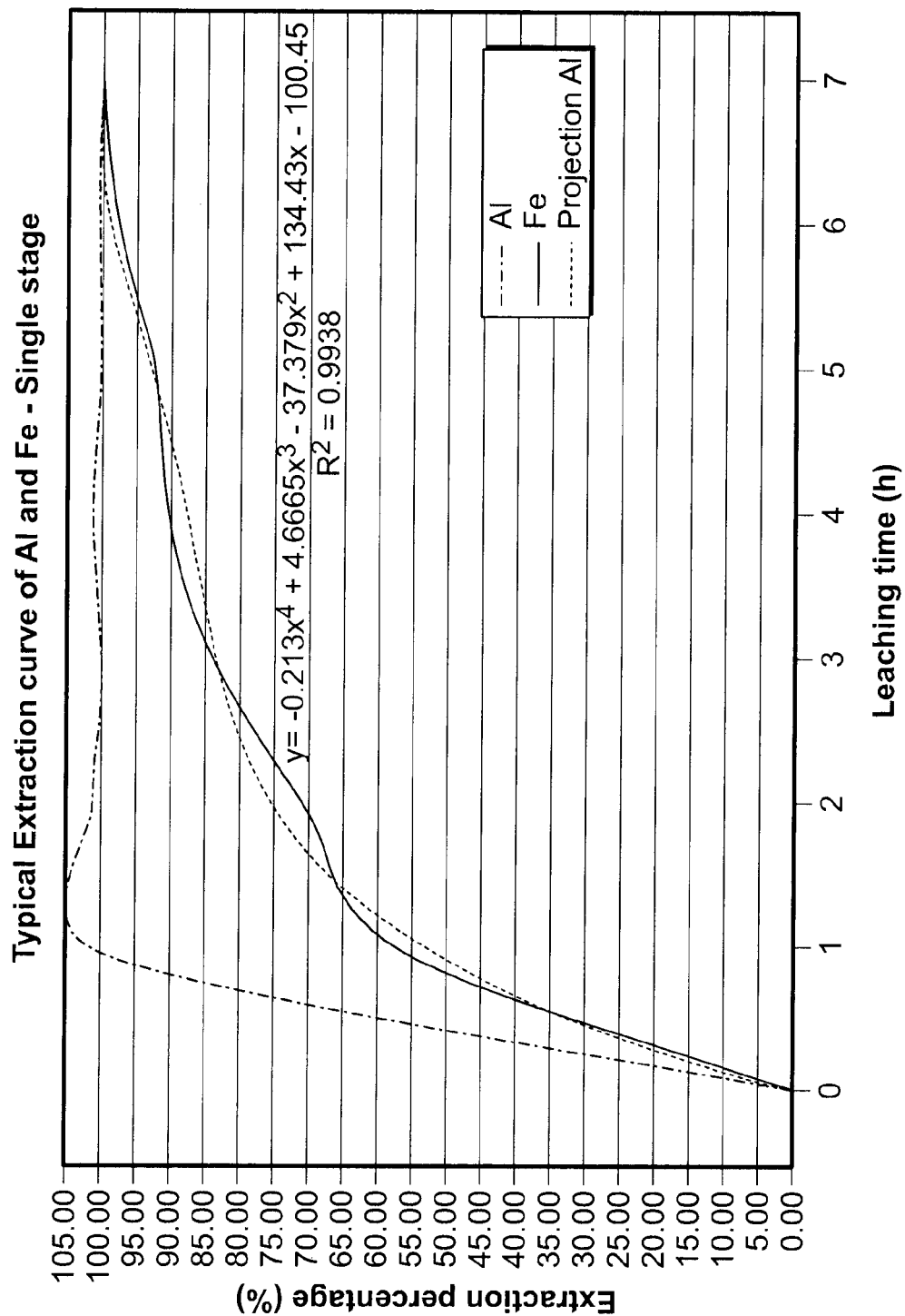
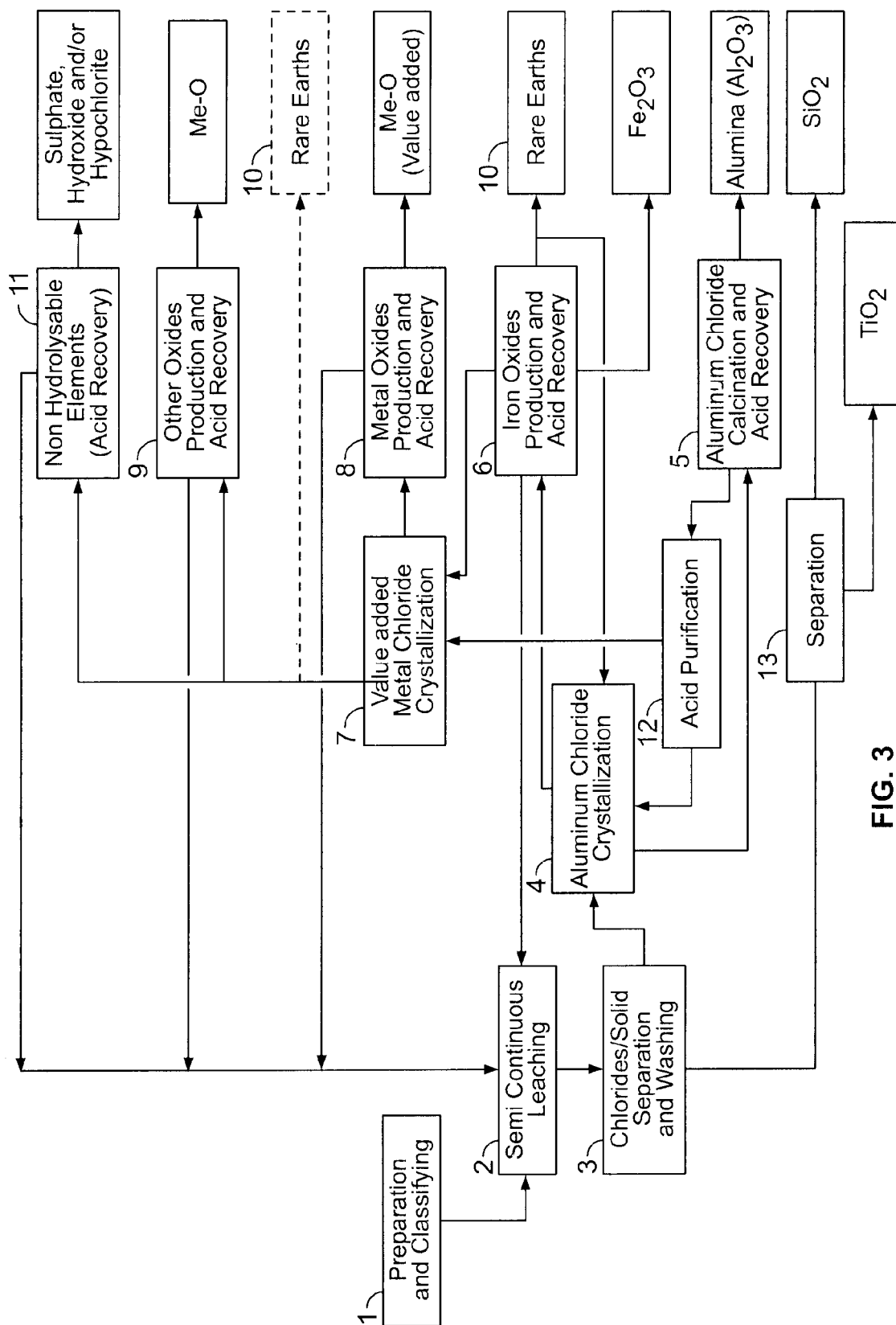


FIG. 2



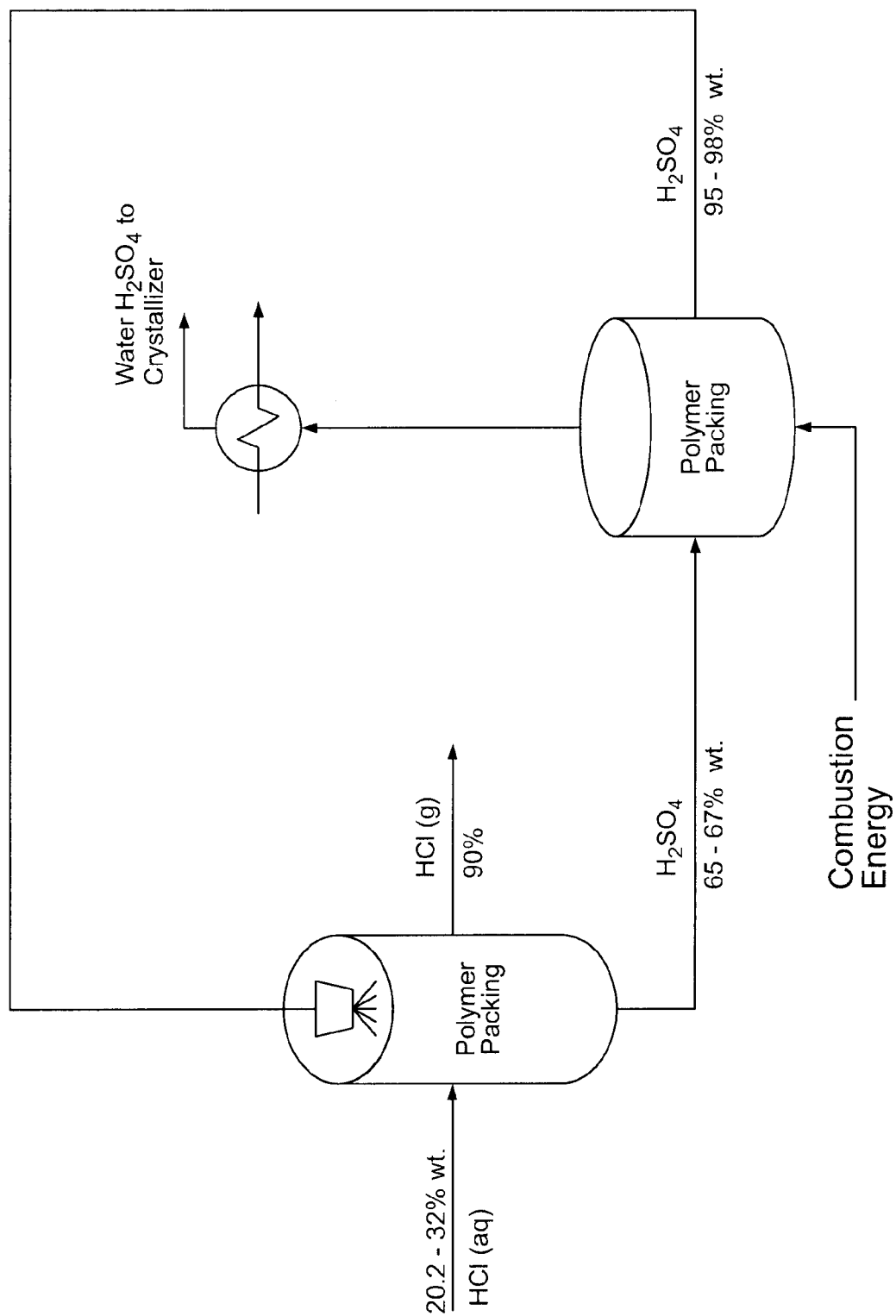


FIG. 4

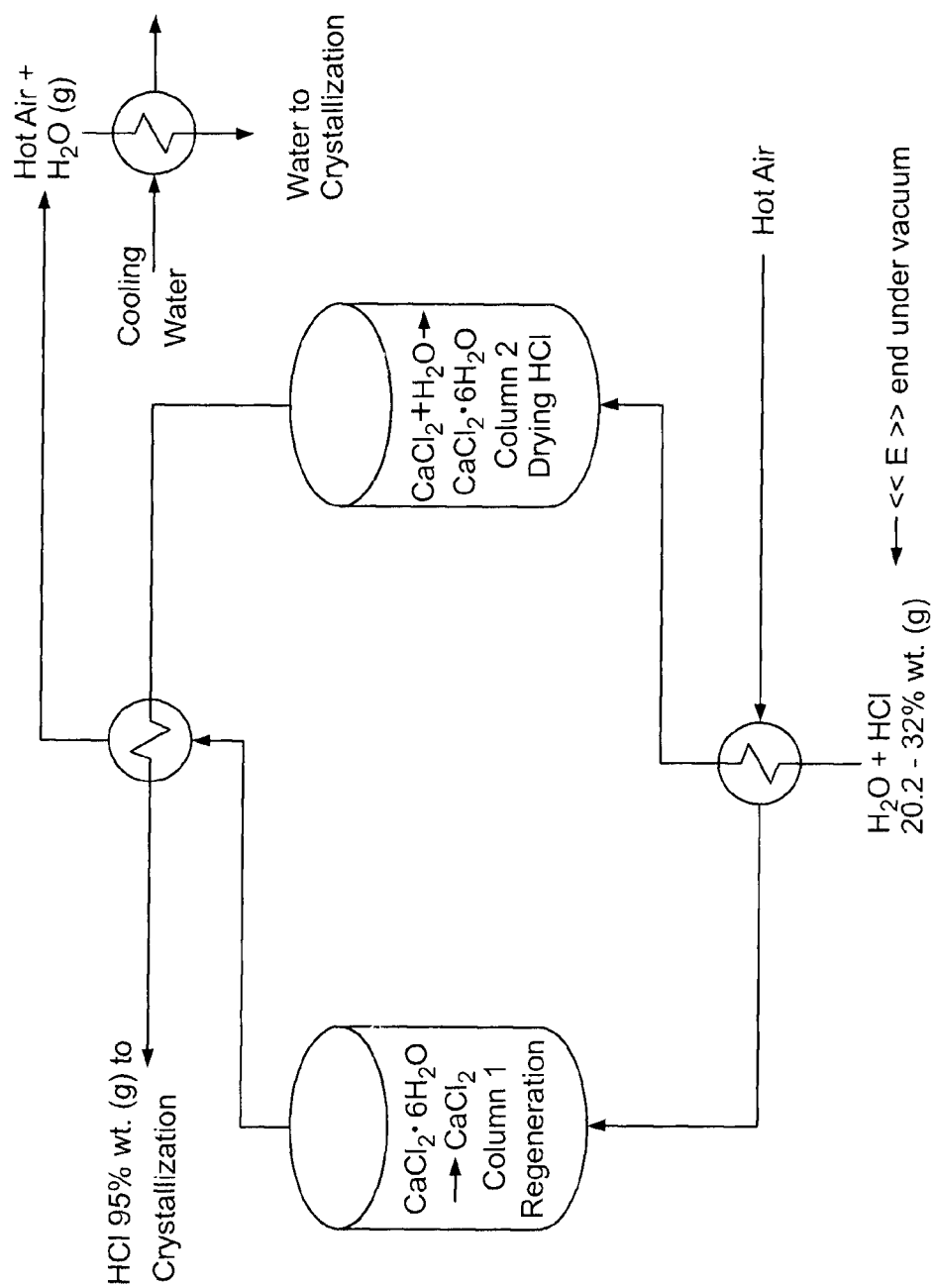
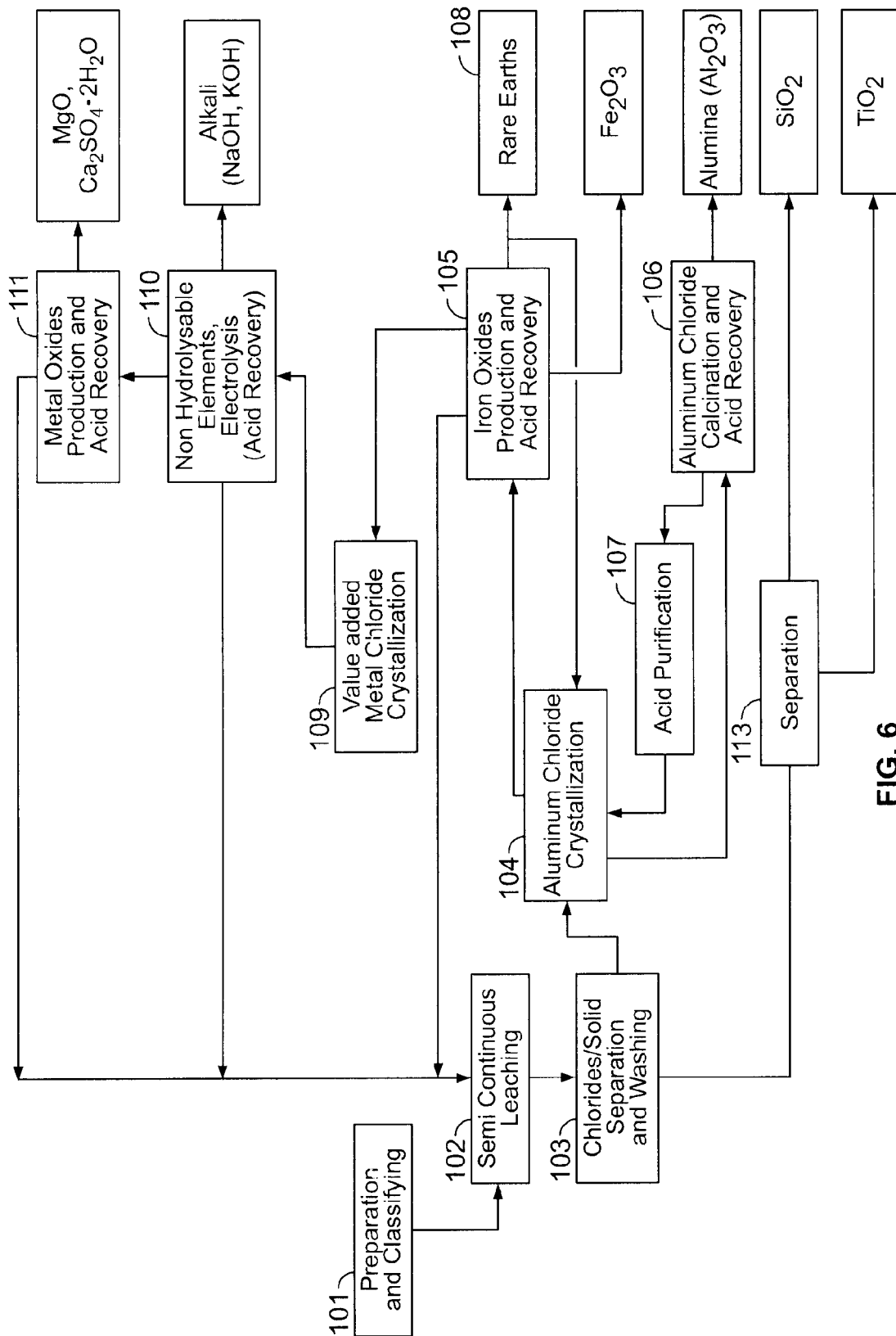


FIG. 5



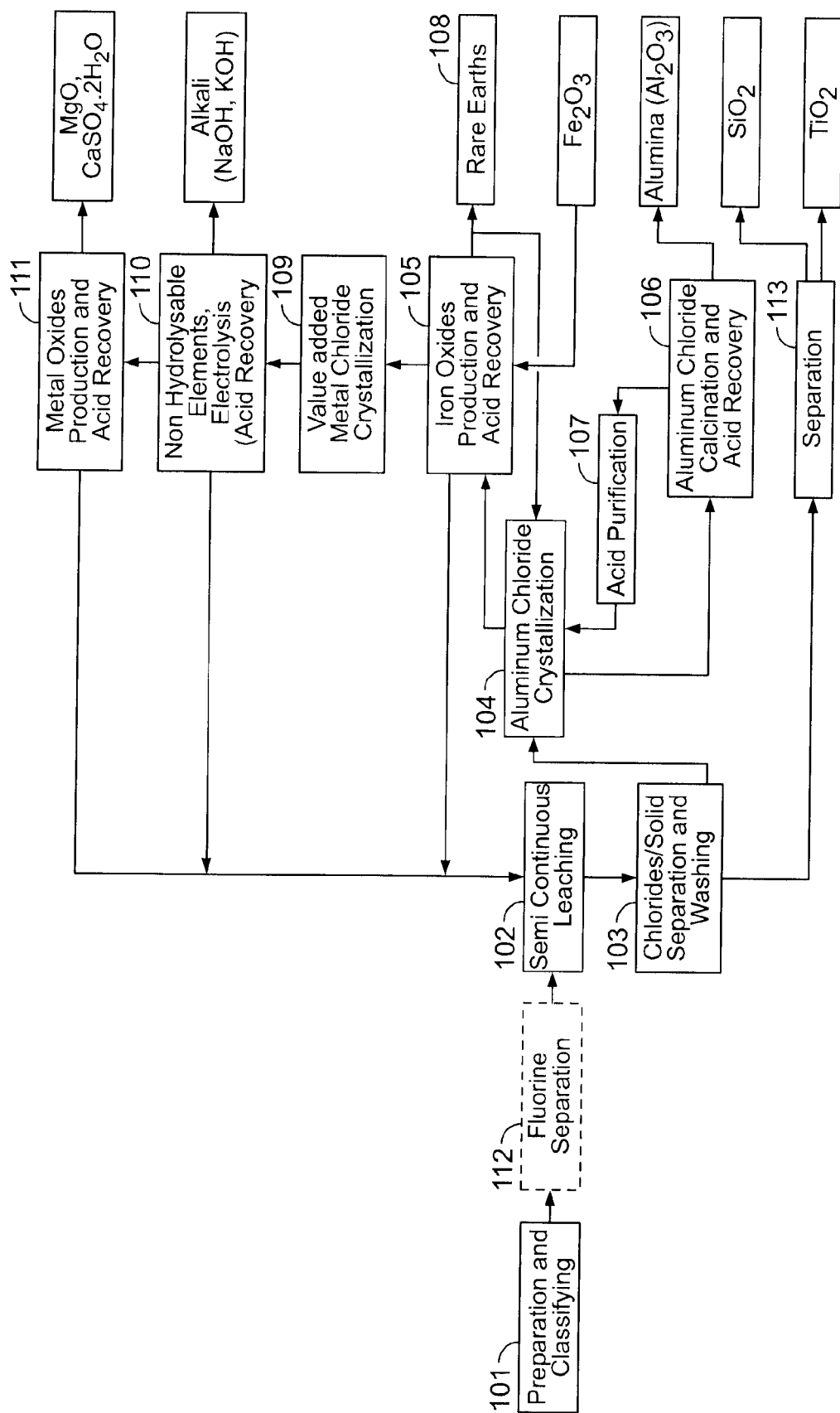


FIG. 7

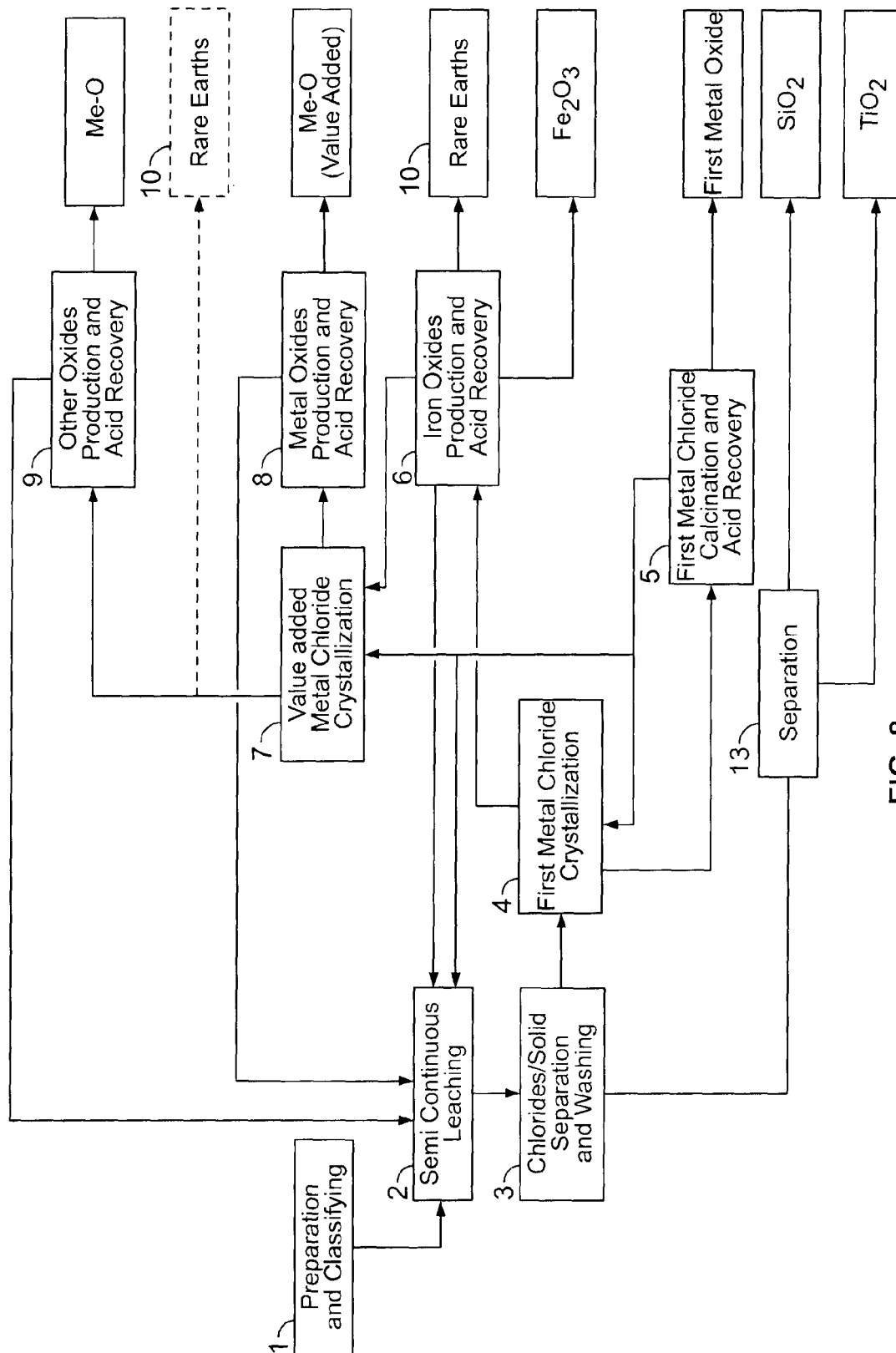


FIG. 8

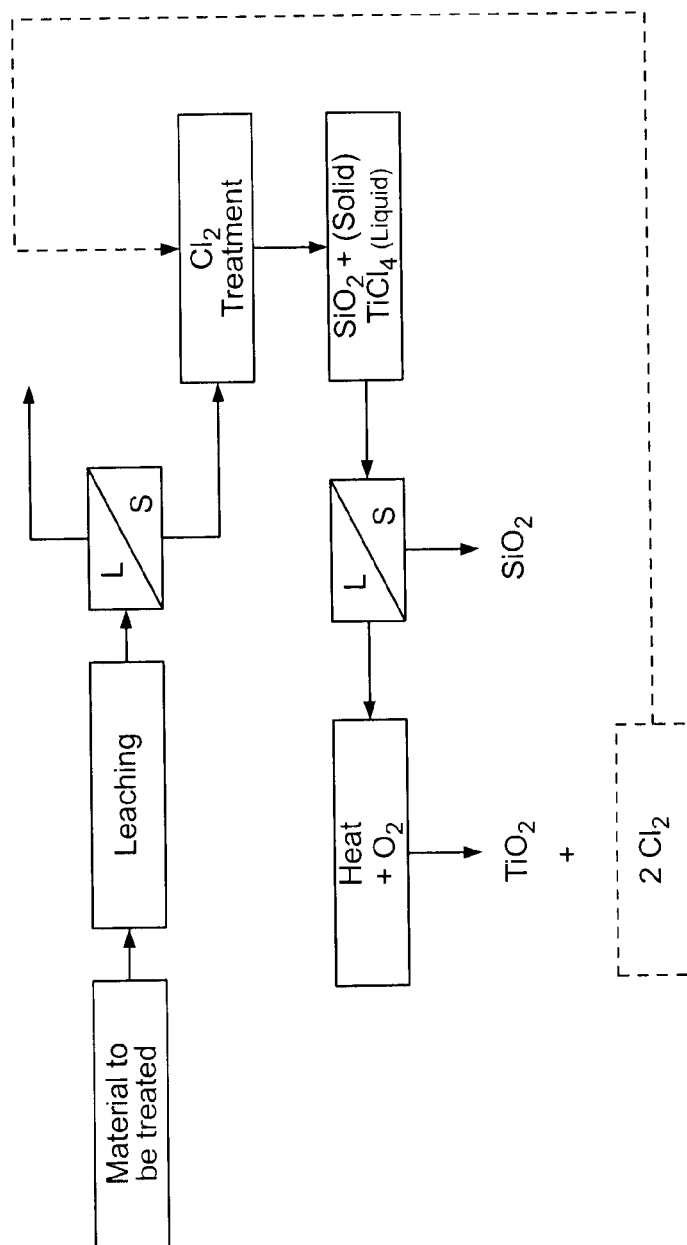


FIG. 9A

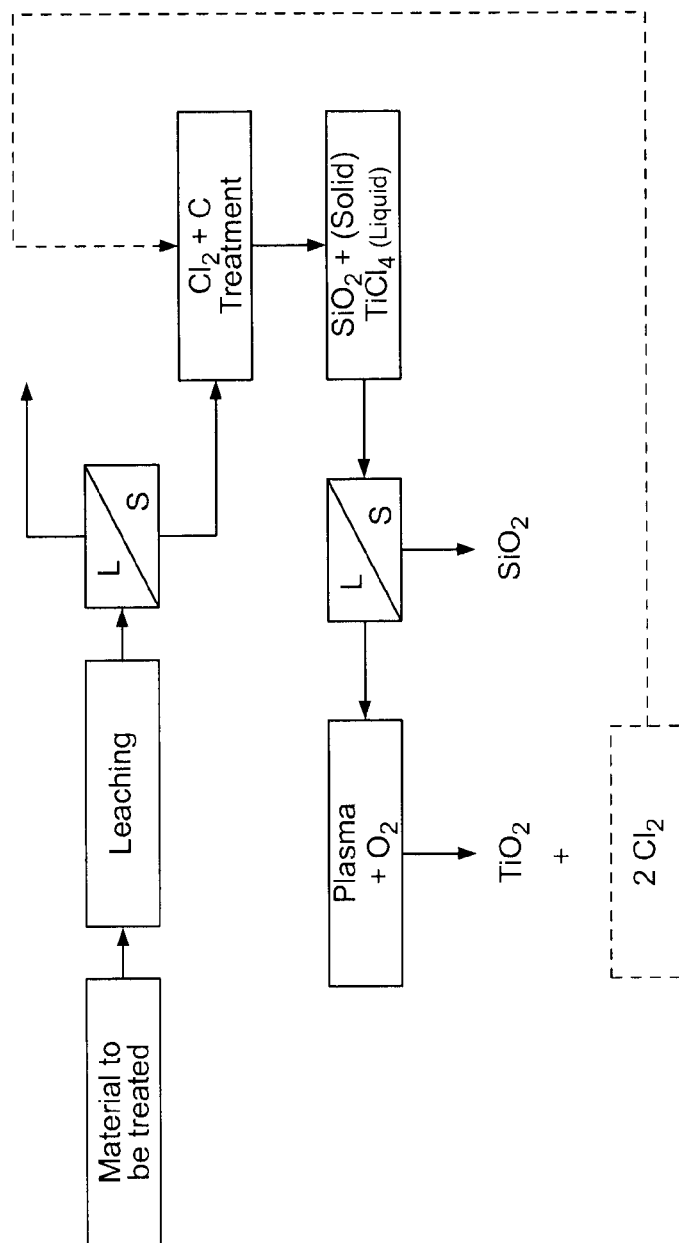


FIG. 9B

1

**PROCESSES FOR PREPARING TITANIUM
OXIDE AND VARIOUS OTHER PRODUCTS****CROSS-REFERENCE TO RELATED
APPLICATIONS**

The present application is a 35 USC 371 national stage entry of PCT/CA2013/000638 filed on Jul. 12, 2013 and which claims priority on U.S. 61/670,645 filed on Jul. 12, 2012; on PCT/CA2012/000871 filed on Sep. 17, 2012; on U.S. 61/726,971 filed on Nov. 15, 2012; and on PCT/CA2013/000021 filed on Jan. 10, 2013. These documents are hereby incorporated by reference in their entirety.

TECHNICAL FIELD

The present disclosure relates to improvements in the field of chemistry applied to the production of alumina and/or titanium oxide. For example, it relates to processes for the production of alumina via the extraction of aluminum from aluminum-containing materials and production of titanium oxide via the extraction of titanium from titanium-containing materials. These processes can also be efficient for preparing other products such as hematite, MgO, silica and oxides of various metals, titanium chloride as well as rare earths and aluminum.

BACKGROUND OF THE DISCLOSURE

There have been several known processes for the production of alumina. Many of them were using bauxite as starting material. These processes, that were mainly alkaline processes, have been employed throughout the years. Several of such alkaline processes have the disadvantage of being inefficient to segregate and extract value added secondary products, thus leaving an important environmental impact. There have also been development work employing hydrochloric acid for the leaching step but, it has been found that such processes were not efficient for removing most part of the impurities and especially iron. Removal of iron was also difficult to be carried out via adequate and economical techniques especially when using continuous processes.

There have also been several processes proposed for preparing titanium oxide but there is still room for improvement.

SUMMARY OF THE DISCLOSURE

According to one aspect, there is provided a process for preparing various products, the process comprising:

leaching a titanium-containing material comprising Ti, Si and a first metal with HCl so as to obtain a leachate comprising ions of the first metal and a solid comprising titanium, and separating the solid from the leachate;

reacting the leachate with HCl so as to obtain a liquid and a precipitate comprising a chloride of the first metal, and separating the precipitate from the liquid;

heating the precipitate under conditions effective for converting the chloride of the first metal into an oxide of the first metal; and

treating the solid so as to substantially separate Si from Ti that are contained therein.

According to another aspect, there is provided a process for preparing alumina, the process comprising:

leaching an aluminum-containing material with HCl so as to obtain a leachate comprising aluminum ions and a solid, and separating the solid from the leachate;

2

reacting the leachate with HCl so as to obtain a liquid and a precipitate comprising the aluminum ions in the form of AlCl_3 , and separating the precipitate from the liquid; heating the precipitate under conditions effective for converting AlCl_3 into Al_2O_3 and optionally recovering gaseous HCl so-produced; and

leaching the solid with HCl optionally in the presence of a chloride so as to obtain a leachate comprising titanium ions.

According to another aspect, there is provided a process for preparing aluminum, the process comprising:

leaching an aluminum-containing material with HCl so as to obtain a leachate comprising aluminum ions and a solid, and separating the solid from the leachate;

reacting the leachate with HCl so as to obtain a liquid and a precipitate comprising the aluminum ions in the form of AlCl_3 , and separating the precipitate from the liquid; heating the precipitate under conditions effective for converting AlCl_3 into Al_2O_3 and optionally recovering gaseous HCl so-produced;

converting the Al_2O_3 into alumina; and

leaching the solid with HCl optionally in the presence of a chloride so as to obtain a leachate comprising titanium ions.

According to another aspect, there is provided a process for preparing alumina and optionally other products, the process comprising:

leaching an aluminum-containing material with HCl so as to obtain a leachate comprising aluminum ions and a solid, and separating the solid from the leachate;

reacting the leachate with HCl so as to obtain a liquid and a precipitate comprising the aluminum ions in the form of AlCl_3 , and separating the precipitate from the liquid; heating the precipitate under conditions effective for converting AlCl_3 into Al_2O_3 and recovering gaseous HCl so-produced;

recycling the gaseous HCl so-produced by contacting it with water so as to obtain a composition having a concentration higher than HCl azeotrope concentration (20.2 weight %) and reacting the composition with a further quantity of aluminum-containing material so as to leaching it; and

leaching the solid with HCl optionally in the presence of a chloride so as to obtain a leachate comprising titanium ions.

According to another aspect, there is provided a process for preparing alumina and optionally other products, the process comprising:

leaching an aluminum-containing material with HCl so as to obtain a leachate comprising aluminum ions and a solid, and separating the solid from the leachate;

reacting the leachate with HCl so as to obtain a liquid and a precipitate comprising the aluminum ions in the form of AlCl_3 , and separating the precipitate from the liquid; heating the precipitate under conditions effective for converting AlCl_3 into Al_2O_3 and recovering gaseous HCl so-produced;

recycling the gaseous HCl so-produced by contacting it with water so as to obtain a composition having a concentration of about 25 to about 45 weight % and reacting the composition with a further quantity of aluminum-containing material so as to leaching it; and

leaching the solid with HCl optionally in the presence of a chloride so as to obtain a leachate comprising titanium ions.

leaching a titanium-containing material with HCl so as to obtain a leachate comprising ions from at least one metal and a solid, and separating the solid from the leachate; at least substantially isolating the ions of the at least one metal from the leachate; and

leaching a titanium-containing material with HCl so as to obtain a leachate comprising ions from at least one metal and a solid, and separating the solid from the leachate; at least substantially isolating the ions of the at least one metal from the leachate;

leaching a titanium-containing material with HCl so as to obtain a first leachate comprising ions from at least one metal and a solid, and separating the solid from the first leachate;

leaching the solid with HCl optionally in the presence of a chloride so as to obtain a second leachate comprising titanium chloride; or reacting the solid with Cl_2 and a carbon source so as to obtain a liquid portion comprising the titanium chloride and a solid portion, and separating the liquid portion from the solid portion; and

leaching a titanium-containing material with HCl so as to obtain a first leachate comprising ions from at least one metal and a solid, and separating the solid from the first leachate;

leaching the solid with HCl optionally in the presence of a chloride so as to obtain a second leachate comprising titanium chloride; and

leaching a titanium-containing material with HCl so as to obtain a first leachate comprising ions from at least one metal and a solid, and separating the solid from the first leachate;

reacting the solid with Cl_2 so as to obtain a liquid portion comprising the titanium chloride and a solid portion, and separating the liquid portion from the solid portion; and converting the titanium chloride into titanium oxide.

leaching a titanium-containing material with HCl so as to obtain a first leachate comprising ions from at least one metal and a solid, and separating the solid from the first leachate;

leaching a titanium-containing material with HCl so as to obtain a first leachate comprising ions from at least one metal and a solid, and separating the solid from the leachate;

reacting the solid with Cl_2 so as to obtain a liquid portion comprising the titanium chloride and a solid portion, and separating the liquid portion from the solid portion.

leaching a titanium-containing material comprising Ti and a first metal with HCl so as to obtain a leachate comprising ions of the first metal and a solid comprising titanium, and separating the solid from the leachate;

treating the solid so as at least partially recover Ti.

leaching a titanium-containing material comprising Ti and Al with HCl so as to obtain a leachate comprising ions of Al and a solid comprising titanium, and separating the solid from the leachate;

treating the solid so as at least partially recover T_i .

leaching a titanium-containing material comprising Ti and Fe with HCl so as to obtain a leachate comprising ions of Fe and a solid comprising titanium, and separating the solid from the leachate;

treating the solid so as at least partially recover T_i .

leaching a titanium-containing material with HCl so as to obtain a first leachate comprising Al ions and a solid, and separating the solid from the first leachate;

leaching the solid with HCl optionally in the presence of a chloride so as to obtain a second leachate comprising titanium chloride; or reacting the solid with Cl_2 so as to obtain a liquid portion comprising the titanium chloride and a solid portion, and separating the liquid portion from the solid portion; and

leaching a titanium-containing material with HCl so as to obtain a first leachate comprising Al ions and a solid, and separating the solid from the first leachate;

at least substantially isolating the Al ions from the first leachate;

leaching the solid with HCl optionally in the presence of a chloride so as to obtain a second leachate comprising titanium chloride; and

5

converting the titanium chloride into titanium oxide.

According to another example, there is provided a process for preparing titanium oxide, the process comprising:

leaching a titanium-containing material with HCl so as to obtain a first leachate comprising Fe ions and a solid, and separating the solid from the first leachate;

at least substantially isolating the Fe ions from the first leachate;

leaching the solid with HCl optionally in the presence of a chloride so as to obtain a second leachate comprising titanium chloride; or reacting the solid with Cl_2 so as to obtain a liquid portion comprising the titanium chloride and a solid portion, and separating the liquid portion from the solid portion; and

converting the titanium chloride into titanium oxide.

According to another example, there is provided a process for preparing titanium oxide, the process comprising:

leaching a titanium-containing material with HCl so as to obtain a first leachate comprising Al ions and a solid, and separating the solid from the first leachate;

at least substantially isolating the Al ions from the first leachate;

leaching the solid with HCl optionally in the presence of a chloride so as to obtain a second leachate comprising titanium chloride; and

converting the titanium chloride into titanium oxide.

According to another example, there is provided a process for preparing titanium oxide, the process comprising:

leaching a titanium-containing material with HCl so as to obtain a first leachate comprising Fe ions and a solid, and separating the solid from the first leachate;

at least substantially isolating the Fe ions from the first leachate;

leaching the solid with HCl optionally in the presence of a chloride so as to obtain a second leachate comprising titanium chloride; and

converting the titanium chloride into titanium oxide.

According to another example, there is provided a process for preparing titanium chloride, the process comprising:

leaching a titanium-containing material with HCl so as to obtain a first leachate comprising ions from at least one metal and a solid, and separating the solid from the first leachate;

at least substantially isolating the ions of the at least one metal from the leachate; and

leaching the solid with HCl optionally in the presence of a chloride so as to obtain a second leachate comprising titanium chloride.

According to another example, there is provided a process for preparing titanium chloride, the process comprising:

leaching a titanium-containing material with HCl so as to obtain a first leachate comprising ions from at least one metal and a solid, and separating the solid from the leachate;

at least substantially isolating the ions of the at least one metal from the first leachate; and

reacting the solid with Cl_2 so as to obtain a liquid portion comprising the titanium chloride and a solid portion, and separating the liquid portion from the solid portion.

According to another aspect, there is provided a process for preparing alumina and optionally other products, the process comprising:

leaching an aluminum-containing material with HCl so as to obtain a leachate comprising aluminum ions and a solid, and separating the solid from the leachate;

6

reacting the leachate with HCl so as to obtain a liquid and a precipitate comprising the aluminum ions in the form of AlCl_3 , and separating the precipitate from the liquid; heating the precipitate under conditions effective for converting AlCl_3 into Al_2O_3 and recovering gaseous HCl so-produced; and

recycling the gaseous HCl so-produced by contacting it with water so as to obtain a composition having a concentration higher than HCl azeotrope concentration (20.2 weight %) and reacting the composition with a further quantity of aluminum-containing material so as to leaching it.

According to another aspect, there is provided a process for preparing alumina and optionally other products, the process comprising:

leaching an aluminum-containing material with HCl so as to obtain a leachate comprising aluminum ions and a solid, and separating the solid from the leachate;

reacting the leachate with HCl so as to obtain a liquid and a precipitate comprising the aluminum ions in the form of AlCl_3 , and separating the precipitate from the liquid; heating the precipitate under conditions effective for converting AlCl_3 into Al_2O_3 and recovering gaseous HCl so-produced; and

recycling the gaseous HCl so-produced by contacting it with water so as to obtain a composition having a concentration of about 25 to about 45 weight % and reacting the composition with a further quantity of aluminum-containing material so as to leaching it.

According to another aspect, there is provided a process for preparing alumina and optionally other products, the process comprising:

leaching an aluminum-containing material with HCl so as to obtain a leachate comprising aluminum ions and a solid, and separating the solid from the leachate;

reacting the leachate with HCl so as to obtain a liquid and a precipitate comprising the aluminum ions in the form of AlCl_3 , and separating the precipitate from the liquid; heating the precipitate under conditions effective for converting AlCl_3 into Al_2O_3 and recovering gaseous HCl so-produced; and

recycling the gaseous HCl so-produced by contacting it with water so as to obtain a composition having a concentration of about 25 to about 45 weight % and using the composition for leaching the aluminum-containing material.

According to another aspect, there is provided a process for preparing alumina and optionally other products, the process comprising:

leaching an aluminum-containing material with HCl so as to obtain a leachate comprising aluminum ions and a solid, and separating the solid from the leachate;

reacting the leachate with HCl so as to obtain a liquid and a precipitate comprising the aluminum ions in the form of AlCl_3 , and separating the precipitate from the liquid; heating the precipitate under conditions effective for converting AlCl_3 into Al_2O_3 and recovering gaseous HCl so-produced; and

recycling the gaseous HCl so-produced by contacting it with the leachate so as to precipitate the aluminum ions in the form of $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$.

According to another aspect, there is provided a process for preparing alumina and optionally other products, the process comprising:

leaching an aluminum-containing material with HCl so as to obtain a leachate comprising aluminum ions and a solid, and separating the solid from the leachate;

7

reacting the leachate with HCl so as to obtain a liquid and a precipitate comprising the aluminum ions in the form of AlCl_3 , and separating the precipitate from the liquid; and

heating the precipitate under conditions effective for converting AlCl_3 into Al_2O_3 .

According to another aspect, there is provided a process for preparing alumina and optionally other products, the process comprising:

leaching an aluminum-containing material with HCl so as to obtain a leachate comprising aluminum ions and a solid, and separating the solid from the leachate;

reacting the leachate with HCl so as to obtain a liquid and a precipitate comprising the aluminum ions in the form of AlCl_3 , and separating the precipitate from the liquid; and

heating the precipitate under conditions effective for converting AlCl_3 into Al_2O_3 and optionally recovering gaseous HCl so-produced.

According to one aspect, there is provided a process for preparing aluminum and optionally other products, the process comprising:

leaching an aluminum-containing material with HCl so as to obtain a leachate comprising aluminum ions and a solid, and separating the solid from the leachate;

reacting the leachate with HCl so as to obtain a liquid and a precipitate comprising the aluminum ions in the form of AlCl_3 , and separating the precipitate from the liquid;

heating the precipitate under conditions effective for converting AlCl_3 into Al_2O_3 ; and converting Al_2O_3 into aluminum.

According to another aspect, there is provided a process for preparing aluminum and optionally other products, the process comprising:

leaching an aluminum-containing material with HCl so as to obtain a leachate comprising aluminum ions and a solid, and separating the solid from the leachate;

reacting the leachate with HCl so as to obtain a liquid and a precipitate comprising the aluminum ions in the form of AlCl_3 , and separating the precipitate from the liquid;

heating the precipitate under conditions effective for converting AlCl_3 into Al_2O_3 and optionally recovering gaseous HCl so-produced; and converting Al_2O_3 into aluminum.

BRIEF DESCRIPTION OF DRAWINGS

In the following drawings, which represent by way of example only, various embodiments of the disclosure:

FIG. 1 shows a bloc diagram of an example of process for preparing alumina and various other products according to the present disclosure;

FIG. 2 is an extraction curve for Al and Fe in which the extraction percentage is expressed as a function of a leaching time in a process according to an example of the present application;

FIG. 3 shows a bloc diagram of another example of process for preparing alumina and various other products according to the present disclosure;

FIG. 4 is a schematic representation of an example of a process for purifying/concentrating HCl according to the present disclosure;

FIG. 5 is a schematic representation of an example of a process for purifying/concentrating HCl according to the present disclosure;

8

FIG. 6 shows another bloc diagram of an example of process for preparing alumina and various other products according to the present disclosure;

FIG. 7 shows another bloc diagram of an example of process for preparing alumina and various other products according to the present disclosure;

FIG. 8 shows another bloc diagram of an example of process for preparing various products; and

FIGS. 9A and 9B show further bloc diagrams of examples of processes according to the present disclosure.

DETAILED DESCRIPTION OF VARIOUS EMBODIMENTS

The following non-limiting examples further illustrate the technology described in the present disclosure.

In the processes of the present disclosure, the material can be, for example, a titanium-containing material.

In the processes of the present disclosure, the material can be, for example, an aluminum-containing material.

In the processes of the present disclosure, the material can be, for example, an iron-containing material.

The aluminum-containing material can be for example chosen from aluminum-bearing ores (such as clays, argillite, mudstone, beryl, cryolite, garnet, spinel, bauxite, or mixtures thereof can be used). The aluminum-containing material can also be a recycled industrial aluminum-containing material such as slag, red mud or fly ashes.

The titanium-containing material can be for example chosen from a titanium-bearing ores (such as ilmenite, rutile, anatase, brookite, perovskite, sphene, eandrewsite, geikieite, pyrophanite or titanomagnetite or mixtures thereof can be used). The titanium-containing material can also be a recycled industrial titanium-containing material such as slag red mud or fly ashes.

For example, the material can be chosen from industrial refractory materials.

For example, the material chosen from aluminosilicate minerals.

The expression "red mud" as used herein refers to an industrial waste product generated during the production of alumina. For example, such a waste product can contain silica, aluminum, iron, calcium, titanium. It can also contains an array of minor constituents such as Na, K, Cr, V, Ni, Ba, Cu, Mn, Pb, Zn etc. For example, red mud can comprises about 15 to about 80% by weight of Fe_2O_3 , about 1 to about 35% by weight Al_2O_3 , about 1 to about 65% by weight of SiO_2 , about 1 to about 20% by weight of Na_2O , about 1 to about 20% by weight of CaO, and up to about 35% by weight of TiO_2 . According to another example, red mud can comprise about 30 to about 65% by weight of Fe_2O_3 , about 10 to about 20% by weight Al_2O_3 , about 3 to about 50% by weight of SiO_2 , about 2 to about 10% by weight of Na_2O , about 2 to about 8% by weight of CaO, and from 0 to about 25% by weight of TiO_2 .

The expression "fly ashes" as used herein refers to an industrial waste product generated in combustion. For example, such a waste product can contain various elements such as silica, oxygen, aluminum, iron, calcium. For example, fly ashes can comprise silicon dioxide (SiO_2) and aluminium oxide (Al_2O_3). For example, fly ashes can further comprises calcium oxide (CaO) and/or iron oxide (Fe_2O_3). For example fly ashes can comprise fine particles that rise with flue gases. For example, fly ashes can be produced during combustion of coal. For example, fly ashes can also comprise at least one element chosen from arsenic, beryllium, boron, cadmium, chromium, chromium VI, cobalt, lead, manganese, mercury,

molybdenum, selenium, strontium, titanium, thallium, and/or vanadium. For example, fly ashes can also comprise rare earth elements. For example, fly ashes can be considered as an aluminum-bearing material.

The expression "slag" as used herein refers to an industrial waste product comprising oxides such as oxides of aluminum, calcium, titanium, magnesium, iron, and/or silicon.

Terms of degree such as "about" and "approximately" as used herein mean a reasonable amount of deviation of the modified term such that the end result is not significantly changed. These terms of degree should be construed as including a deviation of at least $\pm 5\%$ or at least $\pm 10\%$ of the modified term if this deviation would not negate the meaning of the word it modifies.

The expression "titanium chloride" as used herein refers, for example, to a compound chosen from TiCl_2 , TiCl_3 and TiCl_4 and mixtures thereof. For example, it refers to TiCl_4 .

The term "hematite" as used herein refers, for example, to a compound comprising $\alpha\text{-Fe}_2\text{O}_3$, $\gamma\text{-Fe}_2\text{O}_3$, $\beta\text{-FeO.OH}$ or mixtures thereof.

The expression "iron ions" as used herein refers, for example to ions comprising to at least one type of iron ion chosen from all possible forms of Fe ions. For example, the at least one type of iron ion can be Fe^{2+} , Fe^{3+} , or a mixture thereof.

The expression "aluminum ions" as used herein refers, for example to ions comprising to at least one type of aluminum ion chosen from all possible forms of Al ions. For example, the at least one type of aluminum ion can be Al^{3+} .

The expression "at least one aluminum ion", as used herein refers, for example, to at least one type of aluminum ion chosen from all possible forms of Al ions. For example, the at least one aluminum ion can be Al^{3+} .

The expression "at least one iron ion", as used herein refers, for example, to at least one type of iron ion chosen from all possible forms of Fe ions. For example, the at least one iron ion can be Fe^{2+} , Fe^{3+} , or a mixture thereof.

The expression "at least one precipitated iron ion", as used herein refers, for example, to at least one type of iron ion chosen from all possible forms of Fe ions that was precipitated in a solid form. For example, the at least one iron ion present in such a precipitate can be Fe^{2+} , Fe^{3+} , or a mixture thereof.

The expressions "substantially selectively recovering", "substantially selectively recovered" and their equivalents as used herein when referring to recovering a compound refers, for example, to recovering or isolating such a compound together with less than 30, 25, 20, 15, 10, 5, 3, 2 or 1% of impurities. Such impurities can be other compounds such as other metals.

The expressions "substantially selectively precipitating", "substantially selectively precipitate" and their equivalents as used herein when referring to precipitating a compound refers, for example, to precipitating such a compound together with less than 30, 25, 20, 15, 10, 5, 3, 2 or 1% of impurities. Such impurities can be other compounds such as other metals.

For example, the at least one metal can comprise a first metal that is chosen from aluminum, iron, zinc, copper, gold, silver, molybdenum, cobalt, magnesium, lithium, manganese, nickel, palladium, platinum, thorium, phosphorus, uranium and titanium, and/or at least one rare earth element and/or at least one rare metal.

For example, the first metal can be aluminum, iron, or magnesium.

For example, the liquid can comprise a second metal. For example, the liquid can comprise a third metal.

For example, the second metal can be chosen from aluminum, iron, zinc, copper, gold, silver, molybdenum, cobalt, magnesium, lithium, manganese, nickel, palladium, platinum, thorium, phosphorus, uranium and titanium, and/or at least one rare earth element and/or at least one rare metal.

For example, the process can comprise separating Si from Ti that are contained in the solid comprises leaching the solid with an acid so as to obtain a liquid portion comprising Ti and a solid portion containing Si and wherein the liquid portion is separated from the solid portion.

For example, the material can be leached with HCl having a concentration of about 15 to about 45 weight %, of about 20 to about 45 weight %, of about 25 to about 45 weight %, of about 26 to about 42 weight %, of about 28 to about 40 weight %, of about 30 to about 38 weight %, or between 25 and 36 weight %. For example, HCl at about 18 wt % or about 32 wt % can be used.

For example, the material can be leached with HCl having a concentration of about 1 M to about 12 M, about 2 M to about 10 M, about 3 M to about 9 M, about 4 M to about 8 M, about 5 M to about 7 M or about 6 M.

Leaching can also be carried out by adding dry highly concentrated acid (for example, 85%, 90% or 95%) in gas phase into the aqueous solution. Alternatively, leaching can also be carried out by using a weak acid solution (for example <3 wt %).

For example, leaching can be carried out by using HCl having a concentration of about 18 to about 32 wt % in a first reactor and then, by using HCl having concentration of about 90 to about 95% (gaseous) in a second reactor.

For example, leaching can be carried out by using HCl having a concentration of about 18 to about 32 wt % in a first reactor then, by using HCl having concentration of about 90 to about 95% (gaseous) in a second reactor; and by using HCl having concentration of about 90 to about 95% (gaseous) in a third reactor.

For example, leaching can be carried out under an inert gas atmosphere (for example argon or nitrogen).

For example, leaching can be carried out under an atmosphere of NH_3 .

For example, the material can be leached at a temperature of about 125 to about 225° C., about 140 to about 165° C., about 145 to about 160° C., about 150 to about 200° C., about 150 to about 190° C., about 160 to about 190° C., about 185 to about 190° C., about 160 to about 180° C., about 160 to about 175° C., or about 165 to about 170° C.

For example, the material can be leached at a pressure of about 4 to about 10 barg, about 4 to about 8 barg, or about 5 to about 6 barg.

For example, the material can be leached at a pressure of about 50 to about 150 psig, about 60 to about 100 psig, or about 70 to about 80 psig.

For example, the material can be leached with HCl having a concentration of about 10 to about 50 weight %, about 15 to about 45 weight %, of about 18 to about 45 weight % of about 18 to about 32 weight %, of about 20 to about 45 weight %, of about 25 to about 45 weight %, of about 26 to about 42 weight %, of about 28 to about 40 weight %, of about 30 to about 38 weight %, or between 25 and 36 weight %. For example, HCl at about 18 wt % or about 32 wt % can be used.

Leaching can also be carried out by adding dry highly concentrated acid (for example, 85%, 90% or 95%) in gas phase into the aqueous solution. Alternatively, leaching can also be carried out by using a weak acid solution (for example <3 wt %).

For example, leaching can be carried out by using HCl having a concentration of about 18 to about 32 wt % in a first

11

reactor and then, by using HCl having concentration of about 90 to about 95%, or about 95 to about 100% (gaseous) in a second reactor.

For example, leaching can be carried out by using HCl having a concentration of about 18 to about 32 wt % in a first reactor then, by using HCl having concentration of about 90 to about 95% (gaseous) in a second reactor; and by using HCl having concentration of about 90 to about 95% (gaseous) in a third reactor.

For example, leaching can be carried out under an inert gas atmosphere (for example argon or nitrogen).

For example, leaching can be carried out under an atmosphere of NH_3 .

For example a first leaching can be carried out at atmospheric pressure and then, at least one further leaching (for example 1, 2 or 3 subsequent leaching steps) can be carried out under pressure.

For example, the material can be leached at a temperature of about 125 to about 225° C., about 150 to about 200° C., about 160 to about 180° C., or about 165 to about 170° C.

For example, the material can be leached at a temperature of about 125 to about 225° C., about 140 to about 165° C., about 145 to about 160° C., about 150 to about 200° C., about 150 to about 190° C., about 160 to about 190° C., about 185 to about 190° C., about 160 to about 180° C., about 160 to about 175° C., or about 165 to about 170° C.

For example, leaching can be a continuous leaching or semi-continuous.

For example, the processes can further comprise recycling the gaseous HCl so-produced by contacting it with water so as to obtain a composition having a concentration of about 18 to about 45 weight %, about 26 to about 42 weight %, about 25 to about 45 weight %, about 28 to about 40 weight %, about 30 to about 38 weight %, about 18 to about 36%, or about 25 to about 36%.

For example, the processes can further comprise recycling the gaseous HCl so-produced by contacting it with water so as to obtain a composition having a concentration of about 18 to about 45 weight % or about 25 to about 45 weight % and using the composition for leaching the material.

For example, the liquid can comprise iron chloride. Iron chloride can comprise at least one of FeCl_2 , FeCl_3 , and a mixture thereof.

For example, the iron chloride can be hydrolyzed into iron oxide.

For example, iron chloride can be hydrolyzed into hematite.

For example, the liquid can have an iron chloride concentration of at least 30% by weight; and can then be hydrolyzed at a temperature of about 155 to about 350° C.

For example, the liquid can be concentrated to a concentrated liquid having an iron chloride concentration of at least 30% by weight; and then the iron chloride can be hydrolyzed at a temperature of about 155 to about 350° C. while maintaining a ferric chloride concentration at a level of at least 65% by weight, to generate a composition comprising a liquid and precipitated hematite, and recovering the hematite.

For example, non-hydrolysable elements with hematite can be concentrated back to a concentration of about 0.125 to about 52% wt. in circulation loop in view of selective extraction.

For example, the liquid can be concentrated to a concentrated liquid having a concentration of the at least one iron chloride of at least 30% by weight; and then hydrolyzed at a temperature of about 155 to about 350° C.

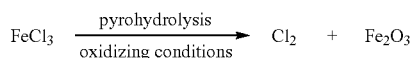
12

For example, the at least one iron chloride can be hydrolyzed at a temperature of about, 150 to about 175, 160 to about 175, 155 to about 170, 160 to about 170 or 165 to about 170° C.

For example, the liquid can be concentrated to a concentrated liquid having a concentration of the at least one iron chloride of at least 30% by weight; and then the at least one iron chloride is hydrolyzed at a temperature of about 155 to about 350° C. while maintaining a ferric chloride concentration at a level of at least 65% by weight, to generate a composition comprising a liquid and precipitated hematite, and recovering the hematite.

For example, the liquid can be concentrated to a concentrated liquid having an iron chloride concentration of at least 30% by weight; and then the iron chloride can be hydrolyzed at a temperature of about 155 to about 350° C. while maintaining a ferric chloride concentration at a level of at least 65% by weight, to generate a composition comprising a liquid and precipitated hematite; recovering the hematite; and recovering rare earths from the liquid.

Alternatively, iron chloride can be treated by pyrohydrolysis so as to generate Cl_2 . For example, such a pyrohydrolysis can be carried out under oxidizing conditions (for example O_2 can be present at a concentration of at least 4, 5, 6, 7, 8, 9, or 10% by weight). Such a reaction can be described as follows:



For example, such a pyrohydrolysis can be carried out in a fluid bed reactor. For example, the Cl_2 so produced can eventually be used for treating the solid comprising Ti.

For example, iron chloride can be pyrohydrolyzed so as to generate Cl_2 .

For example, the Cl_2 so generated can be used for treating said solid and to react with said Ti contained therein.

For example, Cl_2 so generated can be used for treating said solid and to react with TiO_2 contained in said solid, thereby producing TiCl_4 .

For example, said Fe ions can be substantially selectively extracted by converting them into Fe_2O_3 via an hydrolysis.

For example, said Fe ions can be substantially selectively extracted by converting them into Fe_2O_3 via a pyrohydrolysis that allows for generating Cl_2 .

For example, said Fe ions can be substantially selectively isolated by converting them into Fe_2O_3 via a pyrohydrolysis that allows for generating Cl_2 .

For example, said Cl_2 so produced can be effective for treating said solid so as to obtain said liquid portion comprising the titanium chloride.

For example, the process can further comprise, after recovery of the rare earths, reacting the liquid with HCl so as to cause precipitation of MgCl_2 , and recovering same.

For example, the processes can further comprise calcining MgCl_2 into MgO .

For example, the processes can further comprise calcining MgCl_2 into MgO and recycling the gaseous HCl so-produced by contacting it with water so as to obtain a composition having a concentration of about 25 to about 45 weight % or about 18 to about 45 weight % and using the composition for leaching the red mud.

For example, the processes can further comprises, after recovery of the rare earths, reacting the liquid with HCl, and

13

substantially selectively precipitating Na_2SO_4 . For example, Na_2SO_4 can be precipitated by reacting the liquid with H_2SO_4 .

For example, the processes can further comprises, after recovery of the rare earths, reacting the liquid with HCl, and substantially selectively precipitating K_2SO_4 . For example, K_2SO_4 can be precipitated by adding H_2SO_4 .

For example, the liquid can be concentrated to a concentrated liquid having an iron chloride concentration of at least 30% by weight; and then the iron chloride can be hydrolyzed at a temperature of about 155 to about 350° C. while maintaining a ferric chloride concentration at a level of at least 65% by weight, to generate a composition comprising a liquid and precipitated hematite; recovering the hematite; and reacting the liquid with HCl. For example, such a process can further comprises reacting the liquid with H_2SO_4 so as to substantially selectively precipitate K_2SO_4 . The processes can also comprise further reacting the liquid with H_2SO_4 so as to substantially selectively precipitating K_2SO_4 .

For example, the processes can comprise reacting dry individual salts (for example Na or K salts) obtained during the processes with H_2SO_4 and recovering HCl while producing marketable K_2SO_4 and Na_2SO_4 and recovering hydrochloric acid of about 15 to about 90% wt.

For example, sodium chloride produced in the processes can undergo a chemical reaction with sulfuric acid so as to obtain sodium sulfate and regenerate hydrochloric acid. Potassium chloride can undergo a chemical reaction with sulfuric acid so as to obtain potassium sulfate and regenerate hydrochloric acid. Sodium and potassium chloride brine solution can alternatively be the feed material to adapted small chlor-alkali electrolysis cells. In this latter case, common bases (NaOH and KOH) and bleach (NaOCl and KOCl) are produced.

For example, the processes can further comprise, after recovery of the rare earth elements and/or rare metals, recovering NaCl from the liquid, reacting the NaCl with H_2SO_4 , and substantially selectively precipitating Na_2SO_4 .

For example, the processes can further comprise, downstream of recovery of the rare earth elements and/or rare metals, recovering KCl from the liquid, reacting the KCl with H_2SO_4 , and substantially selectively precipitating K_2SO_4 .

For example, the processes can further comprise, downstream of recovery of the rare earth elements and/or rare metals, recovering NaCl from the liquid, carrying out an electrolysis to generate NaOH and NaOCl.

For example, the processes can further comprise, downstream of recovery of the rare earth elements and/or rare metals, recovering KCl from the liquid, reacting the KCl, carrying out an electrolysis to generate KOH and KOCl.

For example, the liquid can be concentrated to a concentrated liquid having a concentration of the at least one iron chloride of at least 30% by weight; and then the at least one iron chloride is hydrolyzed at a temperature of about 155 to about 350° C. while maintaining a ferric chloride concentration at a level of at least 65% by weight, to generate a composition comprising a liquid and precipitated hematite; recovering the hematite; and extracting NaCl and/or KCl from the liquid.

For example, the processes can further comprise reacting the NaCl with H_2SO_4 so as to substantially selectively precipitate Na_2SO_4 .

For example, the processes can further comprise reacting the KCl with H_2SO_4 so as to substantially selectively precipitate K_2SO_4 .

For example, the processes can further comprise carrying out an electrolysis of the NaCl to generate NaOH and NaOCl.

14

For example, the processes can further comprise carrying out an electrolysis of the KCl to generate KOH and KOCl.

For example, the processes can comprise separating the solid from the leachate and washing the solid so as to obtain silica having a purity of at least 95%, at least 96%, at least 97%, at least 98%, at least 99%, at least 99.5% or at least 99.9%.

For example, the process can comprise separating the solid from the leachate and washing the solid so as to obtain silica having a purity of at least 95%.

For example, the processes can comprise reacting the leachate with gaseous HCl so as to obtain the liquid and the precipitate comprising the first metal under the form of a chloride.

For example, the processes can comprise reacting the leachate with dry gaseous HCl so as to obtain the liquid and the precipitate comprising the first metal under the form of a chloride.

For example, precipitating AlCl_3 can comprise crystallizing $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$.

For example, the process can comprise reacting the leachate with HCl recovered during the process and having a concentration of at least 30% as to obtain the liquid and the precipitate comprising the aluminum ions, the precipitate being formed by crystallization of $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$.

For example, the first metal can be chosen from aluminum, iron, zinc, copper, gold, silver, molybdenum, cobalt, magnesium, lithium, manganese, nickel, palladium, platinum, thorium, phosphorus and uranium, and/or at least one rare earth element and/or at least one rare metal.

For example, the second metal can be iron or aluminum.

For example, the liquid can comprise a second metal.

For example, the second metal can be chosen from aluminum, iron, zinc, copper, gold, silver, molybdenum, cobalt, magnesium, lithium, manganese, nickel, palladium, platinum, thorium, phosphorus and uranium and titanium, and/or at least one rare earth element and/or at least one rare metal.

For example, the second metal can be iron or aluminum.

For example, the process can comprise separating the precipitate from the liquid and heating the second metal in order to convert a chloride of the second metal into an oxide of the second metal.

For example, the processes can comprise:

separating the solid from the leachate;

leaching the solid with an acid so as to obtain another leachate; and

recovering a third metal from the another leachate.

For example, the third metal can be chosen from aluminum, iron, zinc, copper, gold, silver, molybdenum, cobalt, magnesium, lithium, manganese, nickel, palladium, platinum, thorium, phosphorus, uranium and titanium, and/or at least one rare earth element and/or at least one rare metal.

For example, the third metal can be titanium.

For example, the acid used for leaching can be chosen from HCl, HNO_3 , H_2SO_4 and mixtures thereof.

For example, the acid can be HCl.

For example, the acid can be gaseous HCl.

For example, the process can comprise recovering the third metal from the another leachate by precipitating the third metal.

For example, the third metal can be precipitated by reacting it with HCl.

For example, the process can further comprise heating the third metal in order to convert a chloride of the third metal into an oxide of the third metal.

For example, the first metal can be aluminum.

For example, the first metal can be magnesium.

15

For example, the first metal can be nickel.

For example, the second metal can be magnesium.

For example, the second metal can be nickel.

For example, the second metal can be iron.

For example, the processes can comprise controlling the temperature of the leachate so as to substantially selectively precipitate the first metal ions in the form of a chloride, and removing the precipitate from the leachate.

For example, the process can comprise reacting the leachate with HCl so as to obtain a liquid and a precipitate comprising a chloride of the first metal, and separating the precipitate from the liquid.

For example, controlling the temperature of the leachate so as to precipitate the first metal in the form of a chloride, and removing the precipitate from the leachate, can be carried out before reacting the leachate with HCl so as to obtain a liquid and a precipitate comprising a chloride of the second metal, and separating the precipitate from the liquid.

For example, controlling the temperature of the leachate so as to precipitate the first metal in the form of a chloride, and removing the precipitate from the leachate, can be carried out after reacting the leachate with HCl so as to obtain a liquid and a precipitate comprising a chloride of the second metal, and separating the precipitate from the liquid.

For example, reacting the leachate with HCl so as to obtain a precipitate comprising the first metal in the form of a chloride, can be carried out by substantially selectively precipitating the first metal chloride.

For example, reacting the leachate with HCl so as to obtain a precipitate comprising the second metal in the form of a chloride, can be carried out by substantially selectively precipitating the second metal chloride.

For example, controlling the temperature of the leachate so as to precipitate the first metal in the form of a chloride can be carried out substantially selectively.

For example, controlling the temperature of the leachate so as to precipitate the second metal in the form of a chloride can be carried out substantially selectively.

For example, controlling the concentration of HCl in the leachate and/or the temperature of the leachate so as to precipitate the first metal in the form of a chloride, can be carried out substantially selectively.

For example, controlling the concentration of HCl in the leachate and/or the temperature of the leachate so as to precipitate the second metal in the form of a chloride, can be carried out substantially selectively.

For example, the processes can comprise reacting the leachate with gaseous HCl so as to obtain the liquid and the precipitate comprising the aluminum ions in the form of $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$.

For example, the processes can comprise reacting the leachate with dry gaseous HCl so as to obtain the liquid and the precipitate comprising the aluminum ions in the form of $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$.

For example, the processes can comprise reacting the leachate with acid of at least 30% wt. that was recovered, regenerated and/or purified as indicated in the present disclosure so as to obtain the liquid and the precipitate comprising the aluminum ions in the form of $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$.

For example, the processes can comprise reacting the leachate with gaseous HCl so as to obtain the liquid and the precipitate comprising the aluminum ions, the precipitate being formed by crystallization of $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$.

For example, the processes can comprise reacting the leachate with dry gaseous HCl so as to obtain the liquid and the precipitate comprising the aluminum ions, the precipitate being formed by crystallization of $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$.

16

For example, aluminum ions can be precipitated under the form of AlCl_3 (for example $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$) in a crystallizer, for example, by adding HCl having a concentration of about 26 to about 32 wt %.

For example, the solid can be treated so as to separate Ti from Si contained therein.

For example, the gaseous HCl can have a HCl concentration of at least 85% wt. or at least 90% wt.

For example, the gaseous HCl can have a HCl concentration of about 90% wt., about 90% to about 95% wt. or about 90% to about 99% wt.

For example, during the crystallization of $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$, the liquid can be maintained at a concentration of HCl of about 25 to about 35% by weight or about 30 to about 32% by weight.

For example, the crystallization can be carried out at a temperature of about 45 to about 65° C. or about 50 to about 60° C.

For example, crystallization of $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$ can be carried out by adding concentrated gaseous HCl to reach a proprietary concentration established of free HCl in the crystalliser. The average results obtained from the crystals For example, the hexahydrate crystals can be fed to the calcination unit. AlCl_3 hydrolysis and conversion can occur at very low temperature (<200° C.). The crystals can pass through a first step where decomposition occurs followed by the calcination itself. The circulating fluid bed can be operated such that energy consumption is less than 30% of the energy normally required for hexahydrate crystal calcination. The alumina produced can be washed to remove unconverted salt if required.

For example, the HCl can be obtained from the gaseous HCl so-produced.

For example, in the processes of the present disclosure, a given batch or quantity of the aluminum-containing material will be leached, will then be converted into AlCl_3 and when the HCl generated during calcination of AlCl_3 into Al_2O_3 will be used for example to leach another given batch or quantity of the aluminum-containing material.

For example, the processes can comprise heating the precipitate at a temperature of at least 180, 230, 250, 300, 350, 400, 450, 500, 550, 600, 650, 700, 750, 800, 850, 900, 925, 930, 1000, 1100, 1200 or 1250° C. for converting AlCl_3 into Al_2O_3 .

For example, converting AlCl_3 into Al_2O_3 can comprise calcination of AlCl_3 .

For example, calcination is effective for converting AlCl_3 into beta- Al_2O_3 .

For example, calcination is effective for converting AlCl_3 into alpha- Al_2O_3 .

For example, converting AlCl_3 into Al_2O_3 can comprise carrying out a calcination via a two-stage circulating fluid bed reactor.

For example, converting AlCl_3 into Al_2O_3 can comprise carrying out a calcination via a two-stage circulating fluid bed reactor that comprises a preheating system.

For example, converting AlCl_3 into Al_2O_3 can comprise carrying out a calcination at low temperature, for example, about 300 to about 600° C., about 325 to about 550° C. about 350 to about 500° C., about 375 to about 450° C., about 375 to about 425° C., or about 385 to about 400° C. and/or injecting steam.

For example, converting AlCl_3 into Al_2O_3 can comprise carrying out a calcination at low temperature, for example, at least 180° C., at least 250° C., at least 300° C., at least 350° C. and/or injecting steam.

17

For example, converting AlCl_3 into Al_2O_3 can comprise carrying out a calcination at low temperature, for example, less than 600°C . and/or injecting steam.

For example, converting AlCl_3 into Al_2O_3 can comprise carrying out a calcination by using coal as combustion source and by using a degasification unit.

For example, the process can comprise converting $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$ into Al_2O_3 by carrying out a calcination of $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$, the calcination comprising steam injection.

For example, steam (or water vapor) can be injected at a pressure of about 200 to about 700 psig, about 300 to about 700 psig, about 400 to about 700 psig, about 550 to about 650 psig, about 575 to about 625 psig, or about 590 to about 610 psig.

For example, steam (or water vapor) can be injected and a plasma torch can be used for carrying fluidization.

For example, the steam (or water vapor) can be overheated.

For example, the steam (or water vapor) can be at a temperature of about 300 to about 400°C .

For example, acid from the offgases generated during calcination can be then treated via a gas phase purification process.

For example, converting AlCl_3 into Al_2O_3 can comprise carrying out a calcination by means of carbon monoxide (CO).

For example, converting AlCl_3 into Al_2O_3 can comprise carrying out a calcination by means of a Refinery Fuel Gas (RFG).

For example, calcination can be carried out by injecting water vapor or steam and/or by using a combustion source chosen from fossil fuels, carbon monoxide, a Refinery Fuel Gas, coal, or chlorinated gases and/or solvents.

For example, calcination can be carried out by injecting water vapor or steam and/or by using a combustion source chosen from natural gas or propane.

For example, calcination can be carried out by providing heat by means of electric heating, gas heating, microwave heating.

For example, the processes can comprise precipitating the Al^{3+} ions under the form of $\text{Al}(\text{OH})_3$. For example, precipitating the Al^{3+} under the form of $\text{Al}(\text{OH})_3$ is carried out at a pH of about 7 to about 10; about 9 to about 10; about 9.2 to about 9.8; about 9.3 to about 9.7; about 9.5; 7.5 to about 8.5; about 7.8 to about 8.2; or about 8.

For example, the iron ions can be precipitated at a pH greater than 11, a pH greater than 12, a pH comprised between 10 and 11, a pH about 11.5 to about 12.5, or a pH about 11.8 to about 12.0.

For example, the Al^{3+} ions can be purified.

For example, the process can comprise precipitating Al^{3+} ions under the form of AlCl_3 so as to purify the Al^{3+} ions. For example, precipitating AlCl_3 can be carried out by crystallizing the AlCl_3 under the form of $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$.

For example, the process can comprise converting AlCl_3 into Al_2O_3 , for example, by converting AlCl_3 into Al_2O_3 under an inert gas atmosphere or by converting AlCl_3 into Al_2O_3 under a nitrogen atmosphere.

The obtained alumina can be washed by demineralized water so as to at least partially remove NaCl and/or KCl.

For example, the fluid bed reactor can comprise a metal catalyst chosen from metal chlorides.

For example, the fluid bed reactor can comprise a metal catalyst that is FeCl_3 , FeCl_2 or a mixture thereof.

For example, the fluid bed reactor can comprise a metal catalyst that is FeCl_3 .

For example, the preheating system can comprise a plasma torch.

18

For example, steam can be used as the fluidization medium heating. Heating can also be electrical.

For example, a plasma torch can be used for preheating the calcination reactor.

For example, a plasma torch can be used for preheating air entering in the calcination reactor.

For example, the plasma torch can be used for generating steam that is injected into a calcination reactor.

For example, the plasma torch can be effective for generating steam that is as fluidization medium in a fluid bed reactor.

For example, a plasma torch can be used for preheating a fluid bed.

For example, the calcination medium can be substantially neutral in terms of O_2 (or oxidation). For example, the calcination medium can favorize reduction (for example a concentration of CO of about 100 ppm).

For example, the calcination medium is effective for preventing formation of Cl_2 .

For example, the processes can comprise converting $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$ into Al_2O_3 by carrying out a calcination of $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$, for example, that is provided by the combustion of gas mixture that comprises:

CH_4 : 0 to about 1% vol;

C_2H_6 : 0 to about 2% vol;

C_3H_8 : 0 to about 2% vol;

C_4H_{10} : 0 to about 1% vol;

N_2 : 0 to about 0.5% vol;

H_2 : about 0.25 to about 15.1% vol;

CO: about 70 to about 82.5% vol; and

CO_2 : about 1.0 to about 3.5% vol.

Such a mixture can be efficient for reduction in off gas volume of 15.3 to 16.3%; therefore the capacity increases of 15.3 to 16.3% proven on practical operation of the circulating fluid bed. Thus for a same flow it represents an Opex of $0.65 \times 16.3\% = 10.6\%$.

For example, the air to natural gas ratio of (Nm^3/h over Nm^3/h) in the fluid bed can be about 9.5 to about 10

For example, the air to CO gas ratio of (Nm^3/h over Nm^3/h) in the fluid bed can be about 2 to about 3.

For example, O_2 can be substantially absent from the gas mixture.

For example, the processes can comprise, before leaching the material, a pre-leaching removal of fluorine optionally contained in the material.

For example, the HCl can be obtained from the gaseous HCl so-produced.

For example, the processes can comprise heating the precipitate at a temperature of at least 1000 , 1100 , 1200 or 1250°C . for converting AlCl_3 into Al_2O_3 .

For example, in the processes of the present disclosure, a given batch or quantity of the material can be leached, will then be converted into AlCl_3 and when the HCl generated during calcination of AlCl_3 into Al_2O_3 will be used for example to leach another given batch or quantity of the aluminum-containing material.

For example, the processes can be effective for providing an Al_2O_3 recovery yield of at least 93%, at least 94%, at least 95%, about 90 to about 95%, about 92 to about 95%, or about 93 to about 95%.

For example, the processes can be effective for providing a Fe_2O_3 recovery yield of at least 98%, at least 99%, about 98 to about 99.5%, or about 98.5 to about 99.5%.

For example, the processes can be effective for providing a MgO recovery yield of at least 96%, at least 97%, at least 98%, or about 96 to about 98%.

For example, the processes can be effective for providing a HCl recovery yield of at least 98%, at least 99%, or about 98 to about 99.9%.

For example, the aluminum-containing material can be argillite.

For example, the aluminum-containing material can be bauxite.

For example, the aluminum-containing material can be red mud.

For example, the processes can be effective for avoiding producing red mud.

For example, the alumina and the other products are substantially free of red mud.

For example, HCl can be recycled. For example, such a recycled HCl can be concentrated and/or purified.

For example, gaseous HCl can be concentrated and/or purified by means of H_2SO_4 . For example, gaseous HCl can be passed through a packed column where it is contacted with a H_2SO_4 countercurrent flow. For example, by doing so, concentration of HCl can be increased by at least 50 wt %, at least 60 wt %, at least 70 wt %, at least 75 wt %, at least 80 wt %, about 50 wt % to about 80 wt %, about 55 wt % to about 75 wt %, or about 60 wt %. For example, the column can be packed with a polymer such as polypropylene or polytrimethylene terephthalate (PTT).

For example, gaseous HCl can be concentrated and/or purified by means of CaCl_2 . For example, gaseous HCl can be passed through a column packed with CaCl_2 .

For example, the processes can further comprise converting alumina (Al_2O_3) into aluminum. Conversion of alumina into aluminum can be carried out, for example, by using the Hall-Héroult process. References is made to such a well known process in various patents and patent applications such as US 20100065435; US 20020056650; U.S. Pat. No. 5,876,584; U.S. Pat. No. 6,565,733. Conversion can also be carried out by means of other methods such as those described in U.S. Pat. No. 7,867,373; U.S. Pat. No. 4,265,716; U.S. Pat. No. 6,565,733 (converting alumina into aluminum sulfide followed by the conversion of aluminum sulfide into aluminum.)

For example, the various products obtained by the processes of the present disclosure such as alumina, hematite, titanium oxides, magnesium oxides, rare earth elements and rare metals, etc can be further purified by means of a plasma torch. For example, they can be individually injected into a plasma torch so as to further purify them.

For example, converting AlCl_3 into Al_2O_3 can comprise carrying out a one-step calcination.

For example, calcination can be carried out at different temperatures with steam. Temperature applied of superheated steam can be of about 350° C. to about 550° C. or about 350° C. to about 940° C. or about 350° C. to about 1200° C.

For example, multi stage evaporation step of the hydrolyser can be carried out to reduce drastically energy consumption.

For example, calcination can be carried out at very low temperature with steam.

For example, multi stage evaporation step of the hydrolyser can be carried out to reduce drastically energy consumption.

For example, leaching of the solid with HCl can be done so as to obtain a leachate comprising titanium ions (for example TiCl_4).

For example, leaching of the solid with HCl can be done in the presence of a chloride so as to obtain a leachate comprising titanium ions (for example TiCl_4). Such a leaching can be carried out as described in U.S. Pat. No. 7,803,336 and WO 2005049872. These documents are hereby integrated by reference in their entirety.

For example, the chloride can be magnesium chloride, calcium chloride or sodium chloride.

For example, the processes, when applicable, can further comprise converting titanium chloride into titanium oxide. Such a conversion can be carried out as described in U.S. Pat. No. 7,803,336 and WO 2005049872. These documents are hereby integrated by reference in their entirety.

For example, the processes can comprise leaching of the material with HCl so as to obtain the leachate comprising aluminum ions and the solid, separating the solid from the leachate; and further treating the solid so as to separate SiO_2 from TiO_2 that are contained therein.

For example, the processes can comprise leaching the material with HCl so as to obtain the leachate comprising aluminum ions and the solid, separating the solid from the leachate; and further treating the solid so as to separate Si from Ti that are contained therein.

For example, the processes can comprise leaching the material with HCl so as to obtain the leachate comprising aluminum ions and the solid, separating the solid from the leachate; and further treating the solid with HCl so as to separate Si from Ti that are contained therein.

For example, the process can comprise leaching the material with HCl so as to obtain the leachate comprising aluminum ions and the solid, separating the solid from the leachate; and further treating the solid with HCl, in the presence of a chloride (for example chosen from alkali chlorides and alkaline earth chlorides), so as to separate Si from Ti that are contained therein.

For example, leaching of the solid with HCl in the presence of a chloride allows for obtaining a leachate comprising titanium chloride.

For example, the solid can be treated with HCl and the chloride so as to obtain a liquid portion comprising Ti and a solid portion containing Si and wherein the liquid portion is separated from the solid portion.

For example, the solid can be treated with HCl and the chloride so as to obtain a liquid portion comprising TiCl_4 .

For example, the process can further comprise converting TiCl_4 into TiO_2 .

For example, TiCl_4 can be converted into TiO_2 by solvent extraction of the third liquid fraction and subsequent formation of titanium dioxide from the solvent extraction.

For example, TiCl_4 can be reacted with water and/or a base to cause precipitation of TiO_2 .

For example, TiCl_4 can be converted into TiO_2 by means of a pyrohydrolysis, thereby generating HCl.

For example, TiCl_4 can be converted into TiO_2 by means of a pyrohydrolysis, thereby generating HCl that is recycled.

For example, TiCl_4 can be converted into TiO_2 by reacting TiCl_4 with O_2 .

For example, TiCl_4 can be converted into TiO_2 by reacting TiCl_4 with O_2 under heating.

For example, HCl generated in the conversion of TiCl_4 into TiO_2 can be recovered.

For example, HCl generated in the conversion of TiCl_4 into TiO_2 can be recovered and reused for leaching the titanium-containing material or the solid

For example, the metal chloride can be MgCl_2 or ZnCl_2 .

For example, the solid can comprise TiO_2 and SiO_2 and the solid can be treated with Cl_2 and carbon in order to obtain a liquid portion and a solid portion, and wherein the solid portion and the liquid portion are separated from one another.

For example, the solid can comprise Ti and Si and the solid can be treated with Cl_2 and carbon in order to obtain a liquid portion and a solid portion, and wherein the solid portion and the liquid portion are separated from one another.

21

For example, the solid can comprise TiO_2 and SiO_2 and the solid can be treated with Cl_2 and carbon in order to obtain a liquid portion and a solid portion, and wherein the solid portion and the liquid portion are separated from one another.

For example, the solid can comprise Ti and Si and the solid can be treated with Cl_2 in order to obtain a liquid portion and a solid portion, and wherein the solid portion and the liquid portion are separated from one another.

For example, the solid can comprise TiO_2 and SiO_2 and the solid can be treated with Cl_2 in order to obtain a liquid portion and a solid portion, and wherein the solid portion and the liquid portion are separated from one another.

For example, the processes can comprise heating titanium chloride so as to convert it into TiO_2 .

For example, the liquid portion can comprise TiCl_2 and/or TiCl_4 .

For example, the liquid portion can comprise TiCl_4 .

For example, the process can further comprise heating TiCl_4 so as to convert it into TiO_2 .

For example, obtained TiO_2 can be purified by means of a plasma torch.

For example, the solid can be treated as described in U.S. Pat. No. 7,182,931 (hereby incorporated by reference in its entirety) in order to obtain TiCl_4 and then converting TiCl_4 into TiO_2 . Regarding the conversion of TiCl_4 into TiO_2 , examples of processes are also provided in U.S. Pat. No. 6,207,131, which is hereby incorporated in its entirety.

For example, the processes can comprise leaching the material with HCl so as to obtain the leachate comprising aluminum ions and the solid, separating the solid from the leachate; and further treating the solid with HCl at a concentration of less than 20% wt., at a temperature of less than 85° C., in the presence of a chloride, so as to separate Si from Ti that are contained therein.

For example, the chloride can be chosen from alkali chlorides and alkaline earth chlorides.

For example, the chloride can be MgCl_2 or CaCl_2 .

After the leaching, the titanium ions under the form of titanium chloride can be in a liquid phase and the Si remains solid. Therefore, Si can thus be simply separated from Ti by a solid/liquid separation. Then, titanium chloride can be converted into TiO_2 . It has to be noted that titanium oxychloride can also be present in the leachate.

Various methods of recovering titanium from a leachate are discussed in CA 2,513,309, which is hereby integrated by reference in its entirety.

For example, separation methods such as solvent extraction, precipitation or ion exchange can be used to remove impurities various impurities e. g. iron, chromium and vanadium, followed by recovery of titanium. Some of these techniques are discussed in the U.S. Pat. No. 6,500,396 (also integrated by reference in its entirety).

For example, in order to purify titanium ions, the leachate can be treated with an organic phase. The organic phase can be selected so that ions of a given can be selectively extracted into the organic phase, with titanium ions remaining in the aqueous solution. Thus, oxides of this given metal can also be obtained in high purity.

Examples of the organic phase are quaternary ammonium chlorides, amines (primary, secondary or tertiary), phosphoric and phosphinic acids, and esters and oxides thereof, e. g. tri-n-butyl phosphate, di-2-ethylhexyl phosphoric acid and phosphine oxide. The organic phase may be stripped from the iron values and recycled. Such an organic phase can be selected so that the titanium chloride can be soluble in the organic phase. For example, the organic phase can be selected such that the organic phase and titanium chloride may be

22

separated by fractional distillation (for example with a separation in boiling points between the organic phase and titanium chloride.)

For example, the titanium chloride can be treated with water or a base, (for example magnesium oxide), or by raising the temperature of the solution to 85-110 C, to effect precipitation of titanium dioxide.

The titanium chloride product obtained can also be subjected to calcination in a pyrohydrolysis reactor or be treated in a plasma torch so as to convert it into TiO_2 .

For example, NaCl recovered from the processes of the present disclosure can be reacted with SO_2 , so as to produce HCl and Na_2SO_4 . Such a reaction that is an exothermic reaction can generate steam that can be used to activate a turbine and eventually produce electricity.

For example, U and/or Th can be treated with the processes of the present disclosure. For example, these two elements can be in such processes in admixtures with iron ions and they can be separated therefrom by means of at least one ion exchange resin.

According to one example as shown in FIG. 1, the processes can involve the following steps (the reference numbers in FIG. 1 correspond to the following steps):

According to one example as shown in FIG. 1, the processes can involve the following steps (the reference numbers in FIG. 1 correspond to the following steps):

1—The aluminum-containing material is reduced to an average particle size of about 50 to about 80 μm .

2—The reduced and classified material is treated with hydrochloric acid which allows for dissolving, under a predetermined temperature and pressure, the aluminum with other elements like iron, magnesium and other metals including rare earth elements and/or rare metals. The silica and titanium (if present in raw material) remain totally undissolved.

3—The mother liquor from the leaching step then undergoes a separation, a cleaning stage in order to separate the solid from the metal chloride in solution.

4—The spent acid (leachate) obtained from step 3 is then brought up in concentration with dry and highly concentrated gaseous hydrogen chloride by sparging this one into a crystallizer. This results into the crystallization of aluminum chloride hexahydrate (precipitate) with a minimum of other impurities. Depending on the concentration of iron chloride at this stage, further crystallization step(s) can be required. The precipitate is then separated from the liquid. For example, particle size of crystals can be about 100 to about 500 microns, about 200 to about 400 microns, or about 200 to about 300 microns. Alternatively, particle size of crystals can be about 100 to about 200 microns, about 300 to about 400 microns or about 400 to 500 microns.

5—The aluminum chloride hexahydrate is then calcined (for example by means of a rotary kiln, fluid bed, etc) at high temperature in order to obtain the alumina form. Highly concentrated gaseous hydrogen chloride is then recovered and excess is brought in aqueous form to the highest concentration possible so as to be used (recycled) in the acid leaching step. Acid can also be directly sent in gas phase to the acid purification stage to increase HCl concentration from about 30 wt % to about 95 wt %. This can be done, for example, during drying stage.

6—Iron chloride (the liquid obtained from step 4) is then pre-concentrated and hydrolyzed at low temperature in view of the Fe_2O_3 (hematite form) extraction and acid recovery from its hydrolysis. All heat recovery from the calcination step (step 5), the leaching part exothermic reaction (step 1) and other section of the processes is being recovered into the

pre-concentrator. Alternatively, the iron chloride can be treated by means of a pyrohydrolysis in order to generate Cl_2 (instead of mainly HCl). In such a case, the Cl_2 so produced can be used in step 13 of FIG. 1 for treating the solid that comprises Ti. For example, it can be used to convert TiO_2 into TiCl_4 . Such a reaction can be carried under oxidizing conditions (for example more than 4% of O_2). The pyrohydrolysis can be carried out in a fluid bed reactor under such oxidizing conditions.

10—After the removal of hematite, a solution rich in rare earth elements and/or rare metals can be processed. As it can be seen in FIG. 3, an internal recirculation can be done (after the removal of hematite) and the solution rich in rare earth elements and/or rare metals can be used for crystallization stage 4. Extraction of the rare earth elements and/or rare metals can be done as described in WO/2012/126092 and/or WO/2012/149642. These two documents are hereby integrated by reference in their entirety.

Other non-hydrolysable metal chlorides (Me-Cl) such as MgCl_2 and others then undergo the following steps:

7—The solution rich in magnesium chloride and other non-hydrolysable products at low temperature is then brought up in concentration with dry and highly concentrated gaseous hydrogen chloride by sparging it into a crystallizer. This results into the precipitation of magnesium chloride as an hexahydrate, for example after sodium and potassium chloride removal.

8—Magnesium chloride hexahydrate is then calcined (either through a rotary kiln, fluid bed, etc.) and hydrochloric acid at very high concentration is thus regenerated and brought back to the leaching step.

9—Other Me-Cl undergo a standard pyrohydrolysis step where mixed oxides (Me-O) can be produced and hydrochloric acid at the azeotropic point (20.2% wt.) is regenerated.

11—Ti contained in the solid obtained from step 3 can be treated so as to separate Si from Ti and thus obtain SiO_2 and TiO_2 .

NaCl produced in this process can undergo chemical reaction with H_2SO_4 to produce Na_2SO_4 and HCl at a concentration at or above azeotropic concentration. Moreover, KCl can undergo chemical reaction with H_2SO_4 to produce K_2SO_4 and HCl having a concentration that is above the azeotropic concentration. Sodium and potassium chloride brine solution can be the feed material to adapted small chlor-alkali electrolysis cells. In this latter case, common bases (NaOH and KOH) and bleach (NaOCl and KOCl) are produced as well as HCl .

For example, the liquid can be concentrated to a concentrated liquid having an iron chloride concentration of at least 30% by weight; and then the iron chloride can be hydrolyzed at a temperature of about 155 to about 350° C. while maintaining a ferric chloride concentration at a level of at least 65% by weight, to generate a composition comprising a liquid and precipitated hematite, and recovering the hematite.

For example, the liquid can be concentrated to a concentrated liquid having an iron chloride concentration of at least 30% by weight; and then the iron chloride can be hydrolyzed at a temperature of about 155 to about 350° C. while maintaining a ferric chloride concentration at a level of at least 65% by weight, to generate a composition comprising a liquid and precipitated hematite; recovering the hematite; and recovering rare earth elements and/or rare metals from the liquid. For example, the process can further comprise, after recovery of the rare earth elements and/or rare metals, reacting the liquid with HCl so as to cause precipitation of MgCl_2 , and recovering same.

As previously indicated, various aluminum-containing materials can be used as starting material of the processes

disclosed in the present disclosure. Examples with clays and bauxite have been carried out. However, the person skilled in the art will understand that the continuous processes can handle high percentages of silica (>55%) and impurities as well as relatively low percentages of aluminum (for example as low as about 15%) and still being economically and technically viable. Satisfactory yields can be obtained (>93-95%) on Al_2O_3 and greater than 75%, 85 or 90% on rare earth elements and/or rare metals. No pre-thermal treatment in most cases are required. The processes disclosed in the present disclosure involve special techniques on leaching and acid recovery at very high strength, thereby offering several advantages over alkaline processes.

In step 1 the mineral, whether or not thermally treated is crushed, milled, dried and classified to have an average particle size of about 50 to about 80 μm .

In step 2, the milled raw material is introduced into the reactor and will undergo the leaching phase.

The leaching hydrochloric acid used in step 2 can be a recycled or regenerated acid from steps 5, 6, 8, 9, 10 and 11 (see FIG. 3) its concentration can vary from 15% to 45% weight. Higher concentration can be obtained using membrane separation, cryogenic and/or high pressure approach. The acid leaching can be carried out under pressure and at temperature close to its boiling point thus, allowing a minimal digestion time and extended reaction extent (90%-100%). Leaching (step 2) can be accomplished in a semi-continuous mode where spent acid with residual free hydrochloric acid is replaced by highly concentrated acid at a certain stage of the reaction or allowing a reduced acid/mineral ratio, thereby reducing reaction time and improving reaction kinetics. For example, kinetic constant k can be: 0.5-0.75 g/mole-L. For example, leaching can be continuous leaching.

As previously indicated, alkali metals, iron, magnesium, sodium, calcium, potassium, rare earth elements and other elements will also be in a chloride form at different stages. Silica and optionally titanium can remain undissolved and will undergo (step 3) a liquid/solid separation and cleaning stage. The processes of the present disclosure tend to recover maximum amount of free hydrochloric acid left and chlorides in solution in order to maximize hydrochloric acid recovery yield, using techniques such as rake classifying, filtration with band filters, centrifugation, high pressure, rotofilters and others. Thanks to step 13, Ti contained in the solid obtained from step 3 can be treated so as to separate Si from Ti and thus obtain SiO_2 and TiO_2 . Various possible strategies can be used to separated Si from Ti as previously indicated. For example, the solid can be further leached (for example with HCl in the presence of a metal chloride (for example MgCl_2 or ZnCl_2) so as to solubilize Ti (for example in the form of TiCl_4) while the Si remains solid. Alternatively, the solid can be reacted with Cl_2 (see FIGS. 9A and 9B). The purified silica can then optionally undergo one or two additional leaching stages (for example at a temperature of about 150 to about 160° C.) so as to increase the purity of silica above 99.9%.

As previously indicated, if iron chloride generated in certain processes of the disclosure is to be treated by means of a pyrohydrolysis in order to generate Cl_2 , the latter can eventually be used in the processes of FIGS. 9A and 9B as a source of Cl_2 .

Pure SiO_2 (one additional leaching stage) cleaning with nano water purity 99% min. Mother liquor free of silica is then named as spent acid (various metal chlorides and water) and goes to the crystallization step (step 4). Free HCl and chlorides recovery can be at least 99, 99.5 or 99.9%.

In step 4, the spent acid (or leachate) with a substantial amount of aluminum chloride is then saturated with dry and

25

highly concentrated gaseous hydrogen chloride obtained or recycled from step 5 or with aqueous HCl >30% wt., which results in the precipitate of aluminum chloride hexahydrate ($\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$). The precipitate retained is then washed and filtered or centrifuged before being fed to the calcination stage (step 5). The remaining of the spent acid from step 4 is then processed to acid recovery system (steps 6 to 8) where pure secondary products will be obtained.

In step 5, aluminum oxide (alumina) is directly obtained from high temperature conditions. The highly concentrated hydrogen chloride in gaseous form obtained can be fed to steps 4 and 7 for crystallization where it can be treated through hydrophobic membranes. The excess hydrogen chloride is absorbed and used as regenerated acid to the leaching step 2 as highly concentrated acid, higher than the concentration at the azeotropic point (>20.2%). For example, such a concentration can be about 18 to about 45 weight %, about 25 to about 45 weight % or between 25 and 36 weight %. Acid can also be redirected in gas phase directly (>30 wt %) to acid purification.

After step 4, various chlorides derivatives (mainly iron with magnesium and rare earth elements and rare metals) are next subjected to an iron extraction step. Such a step can be carried out for example by using the technology disclosed in WO 2009/153321, which is hereby incorporated by reference in its entirety. Moreover, hematite can be seeded for crystal growth. For example, hematite seeding can comprise recirculating the seeding.

In step 6, a hydrolysis at low temperature (155-350° C.) is carried out and pure Fe_2O_3 (hematite) is being produced and hydrochloric acid of at least 15% concentration is being regenerated. The method as described in WO 2009/153321 is processing the solution of ferrous chloride and ferric chloride, possible mixtures thereof, and free hydrochloric acid through a series of steps pre-concentration step, oxidation step where ferrous chloride is oxidized into ferric form, and finally through an hydrolysis step into an operational unit called hydrolyser where the ferric chloride concentration is maintained at 65 weight % to generate a rich gas stream where concentration ensures a hydrogen chloride concentration of 15-20.2% and a pure hematite that will undergo a physical separation step. Latent heat of condensation is recovered to the pre-concentration and used as the heating input with excess heat from the calcination stage (step 5).

The mother liquor from the hydrolyser (step 6) can be recirculated partially to first step crystallization process where an increase in concentration of non-hydrolysable elements is observed. After iron removal, the liquor is rich in other non-hydrolysable elements and mainly comprises magnesium chloride or possible mixture of other elements (various chlorides) and rare earth elements and rare metals that are, for example, still in the form of chlorides.

Rare earth elements and rare metals in form of chlorides are highly concentrated, in percentage, into the hydrolyser operational unit (step 6) and are extracted from the mother liquor (step 10) where various known techniques can be employed to extract a series of individual RE-O (rare earth oxides). Among others, the processes of the present disclosure allows to concentrate to high concentration the following elements, within the hydrolyser: scandium (Sc), gallium (Ga), yttrium (Y), dysprosium (Dy), cerium (Ce), praseodymium (Pr), neodymium (Nd), europium (Eu), lanthanum (La), samarium (Sm), gadolinium (Gd), erbium (Er), zirconium (Zr) and mixtures of thereof. Technologies that can be used for extracting rare earth elements and/or rare metals can be found, for example, in Zhou et al. in RARE METALS, Vol. 27, No. 3, 2008, p 223-227, and in US 2004/0042945, hereby incorpo-

26

rated by reference in their entirety. The person skilled in the art will also understand that various other processes normally used for extracting rare earth elements and/or rare metals from the Bayer process can also be used. For example, various solvent extraction techniques can be used. For certain elements, a technique involving octylphenyl acid phosphate (OPAP) and toluene can be used. HCl can be used as a stripping agent. This can be effective for recovering Ce_2O_3 , Sc_2O_3 , Er_2O_3 etc. For example, different sequence using oxalic acid and metallic iron for ferric chloride separation can be used.

The spent acid liquor from steps 6 and 10 rich in value added metals, mainly magnesium, is processed to step 7. The solution is saturated with dry and highly concentrated gaseous hydrogen chloride from step 5, which results in the precipitation of magnesium chloride hexahydrate. For example, same can be accomplished with HCl in aqueous form over 30% wt. The precipitate retained, is fed to a calcination stage step 8 where pure MgO (>98% wt.) is obtained and highly concentrated hydrochloric acid (for example of at least 38%) is regenerated and diverted to the leaching step (step 2). An alternative route for step 7 is using dry gaseous hydrochloric acid from step 8.

In step 9, metal chlorides unconverted are processed to a pyrohydrolysis step (700-900° C.) to generate mixed oxides and where hydrochloric acid from 15-20.2% wt. concentration can be recovered.

According to another example as shown in FIG. 3, the processes can be similar to the example shown in FIG. 1 but can comprise some variants as below discussed.

In fact, as shown in FIG. 3, the processes can comprise (after step 6 or just before step 10) an internal recirculation back to the crystallization step 4. In such a case, The mother liquor from the hydrolyser (step 6) can be recirculated fully or partially to the crystallization of step 4 where a concentration increase will occur with respect to the non-hydrolysable elements including rare earth elements and/or rare metals.

Such a step can be useful for significantly increasing the concentration of rare earth elements and/or rare metals, thereby facilitating their extraction in step 10.

With respect to step 7, the solution rich in magnesium chloride and other non-hydrolysable products at low temperature is, as previously discussed, then brought up in concentration with dry and highly concentrated gaseous hydrogen chloride by sparging it into a crystallizer. This can result into the precipitation of magnesium chloride as an hexahydrate (for example after sodium and potassium chloride removal). This can also be accomplished with HCl in aqueous form.

As shown in FIG. 3, an extra step 11 can be added. Sodium chloride can undergo a chemical reaction with sulfuric acid so as to obtain sodium sulfate and regenerate hydrochloric acid at a concentration at or above the azeotropic point. Potassium chloride can undergo a chemical reaction with sulfuric acid so as to obtain potassium sulfate and regenerate hydrochloric acid at a concentration above the azeotropic concentration. Sodium and potassium chloride brine solution can be the feed material to adapted small chlor-alkali electrolysis cells. In this latter case, common bases (NaOH and KOH) and bleach (NaOCl and KOCl) are produced and can be reused to some extent in other areas of the processes of the present disclosure (scrubber, etc.).

As previously indicated, if iron chloride is treated in stage 6 of FIG. 3 by means of a pyrohydrolysis under oxidizing conditions, Cl_2 will thus be produced and it can be used for treating the solid of stage 13. In such a case an arrow from box

27

6 towards box 13 can be inserted in FIG. 3 and the arrow from after box 6 to box 4 can be optionally removed.

Similarly, for the processes of FIGS. 6 and 7, if iron chloride is treated in stage 105 by means of a pyrohydrolysis under oxidizing conditions, Cl_2 will thus be produced and it can be used for treating the solid of stage 113. In such a case an arrow from box 105 towards box 113 can be inserted in FIG. 3 and the arrow from after box 105 to box 104 can be optionally removed.

The following are non-limitative examples.

Example 1

Preparation of Alumina and Various Other Products

As a starting material a sample of clay was obtained from the Grande Vallee area in Québec, Canada.

These results represent an average of 80 tests carried out from samples of about 900 kg each.

Crude clay in the freshly mined state after grinding and classification had the following composition:

Al_2O_3 : 15%-26%;

SiO_2 : 45%-50%;

Fe_2O_3 : 8%-9%;

MgO : 1%-2%;

Rare earth elements and/or rare metals: 0.04%-0.07%;

LOI: 5%-10%.

This material is thereafter leached in a two-stage procedure at 140-170° C. with 18-32 weight % HCl. The HCl solution was used in a stoichiometric excess of 10-20% based on the stoichiometric quantity required for the removal of the acid leachable constituents of the clay. In the first leaching stage of the semi-continuous operation (step 2), the clay was contacted for 2.5 hours with required amount or certain proportion of the total amount of hydrochloric acid. After removal of the spent acid, the clay was contacted again with a minimum 18 weight % hydrochloric acid solution for about 1.5 hour at same temperature and pressure.

A typical extraction curve obtained for both iron and aluminum for a single stage leaching is shown in FIG. 2.

The leachate was filtered and the solid was washed with water and analyzed using conventional analysis techniques (see step 3 of FIG. 1). Purity of obtained silica was of 95.4% and it was free of any chlorides and of HCl.

In another example, the purity of the silica was 99.67% through an extra leaching step.

After the leaching and silica removal, the concentration of the various metal chlorides was:

AlCl_3 : 15-20%;

FeCl_2 : 4-6%;

FeCl_3 : 0.5-2.0%;

MgCl_2 : 0.5-2.0%;

REE-Cl: 0.1-2%

Free HCl: 5-50 g/l

Spent acid was then crystallized using about 90 to about 98% pure dry hydrochloric acid in gas phase in two stages with less than 25 ppm iron in the aluminum chloride hexahydrate formed. The concentration of HCl in solution (aqueous phase) was about 22 to about 32% or 25 to about 32%, allowing 95.3% of Al_2O_3 recovery. The recovered crystallized material (hydrate form of AlCl_3 having a minimum purity of 99.8%) was then calcined at 930° C. or 1250° C., thus obtaining the α form of the alumina. Heating at 930° C. allows for obtaining the beta-form of alumina while heating at 1250° C. allows for obtaining the alpha-form.

28

Another example was carried out at low temperature (decomposition and calcination at about 350° C.) and the α form of the alumina was less than 2%.

HCl concentration in gas phase exiting the calcination stage was having a concentration greater than 30% and was used (recycled) for crystallization of the AlCl_3 and MgCl_2 . Excess of hydrochloric acid is absorbed at the required and targeted concentration for the leaching steps.

Iron chloride (about 90-95% in ferric form) is then sent to a hydrothermal process in view of its extraction as pure hematite (Fe_2O_3). This can be done by using the technology described in WO 2009/153321 of low temperature hydrolysis with full heat recovery from calcining, pyrohydrolysis and leaching stage.

Rare earth elements and rare metals are extracted from the mother liquor of the hydrolyzer where silica, aluminum, iron and a great portion of water have been removed and following preconcentration from hydrolyser to crystallization. It was observed that rare earth elements can be concentrated by a factor of about 4.0 to 10.0 on average within the hydrolyzer itself on a single pass through it i.e. without concentration loop. The following concentration factors have been noted within the hydrolyzer (single pass):

Ce>6

La>9

Nd>7

Y>9

Remaining magnesium chloride is sparged with dry and highly concentrated hydrochloric acid and then calcinated to MgO while recovering high concentration acid (for example up to 38.4%).

Mixed oxides (Me-O) containing other non-hydrolysable components were then undergoing a pyrohydrolysis reaction at 700-800° C. and recovered acid (15-20.2% wt.) was rerouted for example to the leaching system.

Overall Yields Obtained:

Al_2O_3 : 93.0-95.03% recovery;

Fe_2O_3 : 92.65-99.5% recovery;

Rare earth elements: 95% minimum recovery (mixture);

MgO : 92.64-98.00% recovery;

Material discarded: 0-5% maximum;

HCl global recovery: 99.75% minimum;

HCl strength as feed to leaching 15-32% (aqueous); 95% (gas)

Red mud production: none.

Example 2

Preparation of Alumina and Various Other Products

A similar feed material (bauxite instead of clay) was processed as per in example 1 up to the leaching stage and revealed to be easily leachable under the conditions established in example 1. It provided an extraction percentage of 100% for the iron and over 90-95% for aluminum. The technology was found to be economically viable and no harmful by-products (red mud) were generated. Samples tested had various concentrations of Al_2O_3 (up to 51%), Fe_2O_3 (up to 27%) and MgO (up to 1.5%). Gallium extraction of 97.0% was observed. Scandium extraction was 95%.

Example 3

HCl Gas Enrichment and Purification

H_2SO_4 Route

H_2SO_4 can be used for carrying out purification of HCl. It can be carried out by using a packing column with H_2SO_4

29

flowing counter currently (see FIG. 4). This allows for converting the recovered HCl into HCl having a concentration above the azeotropic point (20.1% wt) and increase its concentration by about 60 to about 70% at minimum.

Water is absorbed by H_2SO_4 and then H_2SO_4 regeneration is applied where H_2SO_4 is brought back to a concentration of about 95 to about 98% wt. Water release at this stage free of sulphur is recycled back and used for crystallization dissolution, etc. Packing of the column can comprise polypropylene or polytrimethylene terephthalate (PTT).

Combustion energy can be performed with off gas preheating air and oxygen enrichment. Oxygen enrichment: +2% represents flame temperature increase by: 400° C. maximum.

Thus, HCl of the processes of the present disclosure can thus be treated accordingly.

Example 4

HCl Gas Enrichment and Purification

Calcium Chloride to Calcium Chloride Hexahydrate (Absorption/Desorption Process)

As shown in FIG. 5, CaCl_2 can be used for drying HCl. In fact, CaCl_2 can be used for absorbing water contained into HCl. In such a case, CaCl_2 is converted into its hexachloride form ($\text{CaCl}_2 \cdot 6\text{H}_2\text{O}$) and one saturated system is eventually switched into regeneration mode where hot air recovered from calcination off gas of alumina and magnesium oxide spray roasting is introduced to regenerate the fixed bed. Alternatively, other absorbing agent such as LiCl can be used instead of CaCl_2 . Such an ion/exchange type process can be seen in FIG. 4 and the cycle can be inversed to switch from one column to another one.

The person skilled in the art would understand that the processes described in examples 3 and 4 (see FIGS. 4 and 5) can be used in various different manners. For example, these processes can be combined with the various processes presented in the present disclosure. For example, such purification techniques can be integrated to the processes shown in FIGS. 1, 3 and 6 to 8. For example, these techniques can be used downstream of at least one of step chosen from steps 5, 6, 8, 9, 10, 11, 13 and 20 (see FIGS. 1, 3 and 8). They can also be used downstream of step 4 and/or step 7. They can also be used downstream of at least one of step chosen from steps 104 to 111 (see FIGS. 6 and 7).

Example 5

Preparation of Alumina and Various Other Products

This example was carried out by using a process as represented in FIGS. 6 and 7. It should be noted that the processes represented in FIGS. 6 and 7 differ mainly by the fact that FIG. 7 shows an extra stage i.e. stage 112.

Raw Material Preparation

Raw material, clay for example, was processed in a secondary crusher in the clay preparation plant 101. Dry milling and classifying occurs on a dry basis in vertical roller mills (for example Fuller-Loesche LM 30.41). The clay preparation 101 included three roller mills; two running at a capacity of approximately 160-180 tph and one on standby. Raw material, if required, can be reduced to 85% less than 63 microns. Processed material was then stored in homogenization silos before being fed to the acid leaching plant 102. Below in Table 1 are shown results obtained during stage 101. If the ore contains the fluorine element, a special treatment can be

30

applied before carrying out the 102 stage. In presence of hydrochloric acid, fluorine can produce hydrofluoric acid. This acid is extremely corrosive and damaging for human health. Thus, before leaching 102, an optional treatment fluorine separation 112 can be done. Stage 112 can comprise treating the processed material coming from stage 101 with an acid in a pre-leaching treatment so as to remove hydrofluoric acid. Therefore, depending on the composition of the raw material, a fluorine separation stage 112 (or pre-leaching stage 112) can be carried out.

TABLE 1

Clay preparation			
Rate	290 tph		
Composition feed (main constituents)	SiO_2 :	50.9%	
	Al_2O_3 :	24.0%	
	Fe_2O_3 :	8.51%	
	CaO :	0.48%	
	MgO :	1.33%	
	Na_2O :	1.06%	
	K_2O :	2.86%	
	MnO :	0.16%	
	Cr_2O_3 :	0.01%	
	TiO_2 :	0.85%	
	P_2O_5 :	0.145%	
	SrO :	0.015%	
	BaO :	0.05%	
	V_2O_5 :	0.0321%	
	Other (including H_2O and REE):	9.63%	
Obtained particle size	85% < 63 μm		
Residual moisture	0.5-0.7%		
Yield	99.5% min		

Acid Leaching

Next, acid leaching 102 was performed semi-continuously in an 80 m³ glass-lined reactor. Semi-continuous mode comprises replacing reacted acid 1/3 in the reaction period with higher concentration regenerated acid, which greatly improves reaction kinetics. The reactor arrangement comprises for example, a series of three reactors. Other examples have been carried out with a first leaching at 1 atm was carried out and then, a second and third semi-continuous or continuous leaching was carried out with aqueous or gaseous HCl.

Leaching was performed at high temperature and pressure (about 160 to about 195° C. and pressures of about 5 to about 8 barg) for a fixed period of time. Reaction time was a function of the reaction extent targeted (98% for Al_2O_3), leaching mode, acid strength, and temperature/pressure applied.

Spent acid recovered out of the acid leaching 102 was then filtered 103 from unreacted silica and titanium dioxide and washed through an automated filter press where all free HCl and chloride are recovered. Step 113 can then be carried out in various manners as indicated previously for step 13. This allows, for example, a maximum quantity of about 30 ppm SiO_2 going into spent liquor. Cleaned silica at a concentration of $\approx 96\% + \text{SiO}_2$ is then produced. Various options are possible at that point. For example, the 96% silica can undergo final neutralization through caustic bath, cleaning, and then brick-eting before storage. According to another example, the silica purified by adding another leaching step followed by a solid separation step that ensures TiO_2 removal (see stage 113 in FIGS. 6 and 7). In that specific case, high purity silica 99.5%+ is produced. In stage 113, titanium and silicium can be separated from one another in various manners. For example, the solid obtained from stage 103 can be leached in the presence of MgCl_2 at a temperature below 90 or 80° C. and at low acid concentration. For example, acid concentration can be below 25 or 20%. The acid can be HCl or H_2SO_4 . In such a case,

31

titanium remains soluble after such a leaching while titanium is still in a solid form. The same also applies when the solid is treated with Cl_2 . These solid and liquid obtained after stage 113 are thus separated to provide eventually TiO_2 and SiO_2 . Water input and flow for silica cleaning is in a ratio of 1:1 (silica/water) (150 t/h SiO_2 /150 t/h H_2O), but comprises of wash water circulation in closed loop in the process and limited amount of process water for final cleaning of the silica and recovery of all chlorides and free HCl generated at the leaching stage. Below in Table 2 are shown results obtained during stage 102.

TABLE 2

Acid Leaching		
Equivalent solid feed rate	259.6 tph	
Operation mode	Semi-continuous	
Acid to clay ratio	3.10 @ 23% wt (Equivalent to 3.35 with semi-continuous at 18.0% wt)	
Regenerated acid concentration	18.0-32.0%	
Operating temperature	150-155° C. (Pilot) 165-200° C. (Plant)	
MAWP	120 psig	
Typical chemical reactions	$\text{Fe}_2\text{O}_3 + 6\text{HCl} \rightarrow 2\text{FeCl}_3 + 3\text{H}_2\text{O}$ $\text{Al}_2\text{O}_3 + 6\text{HCl} \rightarrow 2\text{AlCl}_3 + 3\text{H}_2\text{O}$ $\text{MgO} + 2\text{HCl} \rightarrow \text{MgCl}_2 + \text{H}_2\text{O}$ $\text{K}_2\text{O} + 2\text{HCl} \rightarrow 2\text{KCl} + \text{H}_2\text{O}$ $\text{Re}_2\text{O}_3 + 6\text{HCl} \rightarrow 2\text{ReCl}_3 + 3\text{H}_2\text{O}$	
Spent acid flow to crystallization	600-1100 m^3/h	
Practical chemical composition after step 102 without solid (SiO_2)	FeCl_3 4.33% FeCl_2 0.19% AlCl_3 16.6% MgCl_2 0.82% NaCl 1.1% KCl 1.2% CaCl_2 0.26% Iron 100% Al_2O_3 98%	
Extraction yields		
SiO_2 Recovery	99.997%	
Energy consumption	Activation energy only and self-sustained exothermic reaction from 130° C.	

AlCl_3 Crystallization

Spent acid, with an aluminum chloride content of about 20 to about 30%, was then processed in the crystallization stage 104. Dry and highly concentrated HCl (>90% wt.) in gas phase was sparged in a two-stage crystallization reactor, which allows the crystallization of aluminum chloride hexahydrate.

The flow rate of acid through these reactors is about 600 to about 675 m^3/h and the reactor was maintained at about 50 to about 60° C. during this highly exothermic reaction. Heat was recovered and exchanged to the acid purification 107 part of the plant thus ensuring proper heat transfer and minimizing heat consumption of the plant. Aluminum chloride solubility decreases rapidly, compared to other elements, with the increase in concentration of free HCl in the crystallization reactor. The concentration of AlCl_3 for precipitation/crystallization was about 30%

The HCl concentration during crystallization was thus about 30 to about 32% wt.

The aqueous solution from the crystallization stage 104 was then submitted to the hydrothermal acid recovery plant 105, while the crystals are processed through the decomposition/calcination stage in the calcination plant 106.

A one-step crystallization stage or a multi-step crystallization stage can be done. For example, a two-steps crystallization stage can be carried out.

32

Below in Tables 3A and 3B are shown results obtained during stage 104.

TABLE 3A

Aluminum chloride crystallization	
Number of crystallization steps	2
Operating temperature	50-60° C.
Sparging HCl concentration	90% (gaseous)
Typical chemicals formed	$\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$ (s)
	Metal chlorides (aq)
$\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$ residual	<5% (practical); 8%

TABLE 3B

Typical crystals composition main constituents obtained at pilot scale and feeding calcination		
Component	Weight distribution (%)	
$\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$	99.978	
$\text{BaCl}_2 \cdot 2\text{H}_2\text{O}$	0.0000	
$\text{CaCl}_2 \cdot 6\text{H}_2\text{O}$	0.0009	
CrCl_4	0.0022	
$\text{CuCl}_2 \cdot 2\text{H}_2\text{O}$	0.0000	
$\text{FeCl}_3 \cdot 6\text{H}_2\text{O}$	0.0019	
KCl	0.0063	
$\text{MgCl}_2 \cdot 6\text{H}_2\text{O}$	0.0093	
$\text{MnCl}_2 \cdot 4\text{H}_2\text{O}$	0.0011	
NaCl	0.0021	
SiCl_4	0.0004	
$\text{SrCl}_2 \cdot 6\text{H}_2\text{O}$	0.0000	
TiCl_4	0.0001	
VCl_4	0.0000	
Free Cl^-	0.0000	

Calcination and Hydrothermal Acid Recovery

The calcination 106 comprises the use of a two-stage circulating fluid bed (CFB) with preheating systems. The pre-heating system can comprise a plasma torch to heat up steam to process. It processes crystals in the decomposition/calcination stage. The majority of the hydrochloric acid was released in the first stage which was operated at a temperature of about 350° C., while the second stage performs the calcination itself. Acid from both stages (about 66 to about 68% of the recovered acid from the processes) was then recovered and sent to either to the acid leaching 102 or to the acid purification 107. In the second reactor, which was operated at a temperature of about 930° C., acid was recovered through the condensation and absorption into two columns using mainly wash water from the acid leaching sector 102. Latent heat from this sector was recovered at the same time as large amounts of water, which limits net water input.

In the iron oxides productions and acid recovery 105 system, which comprises, aqueous solution from the crystallization 104 first undergoes a pre-concentration stage followed by processing in the hydrolyzer reactor. Here, hematite was produced during low temperature processing (about 165° C.). A recirculation loop was then taken from the hydrolyzer and is recirculated to the pre-concentrator, allowing the concentration of REE, Mg, K, and other elements. This recirculation loop, allows rare earth element chlorides and/or rare metal chlorides and various metal chlorides concentration to increase without having these products precipitating with hematite up to a certain extent.

Depending on acid balance in the plant, recovered acid is sent either directly to the 102 or 107 stage. Table 4 shows results obtained in stage 105.

33

TABLE 4

Hydrothermal acid recovery	
Flowrate from crystallization to HARP	592 m ³ /h (design) 600 m ³ /h (design)
Operating hydrolyser temperature	155-170° C.
Regenerated acid concentration	27.4%
Regenerated acid flowrate	205.2 tph HCl
Hematite total production rate	24 TPH (design)
HCl recovery	<99.8%
Reflux (recirculation loop) rate in between hydrolyzer and pre-concentrator	56 tph
Rare earth element chlorides and/or rare metal chlorides rate in recirculation loop	≈12.8 t/h
Hematite quality obtained and/or projected	
Fe ₂ O ₃ purity	>99.5%
Hydrolysable chlorides	<0.2%
Moisture	Max 20% after filtration
PSD	25-35 microns
Density (bulk)	2-3 kg/l
Typical chemical reaction in stage 105	
$2\text{FeCl}_3 + 3\text{H}_2\text{O} \rightarrow \text{Fe}_2\text{O}_3 + 6\text{HCl}$ 155-170° C.	

Table 5 shows results obtained in stage 106.

TABLE 5

Calcination Plant 106	
Process characteristics:	Two-stage circulating fluid bed (CFB) with pre-heating system Two-stage hydrochloric acid regeneration
Production rate (practical)	About 66 tph
CFB feed rate	371 tph @ 2-3% humidity*
Typical chemical reaction occurring	
$2(\text{AlCl}_3 \cdot 6\text{H}_2\text{O}) + \text{Energy} \rightarrow \text{Al}_2\text{O}_3 + 6\text{HCl} + 9\text{H}_2\text{O}$	
Typical alumina chemical composition obtained from aluminum chloride hexahydrate crystals being fed to calcination	
Component	Weight distribution (%)
Al ₂ O ₃	99.938
Fe ₂ O ₃	0.0033
SiO ₂	0.0032
Cr ₂ O ₃	0.0063
V ₂ O ₅	0.0077
Na	0.0190
MgO	0.0090
P ₂ O ₅	0.0039
K	0.0053
Ca	0.0020
MnO	0.0002
Free Cl ⁻	Undetectable

Rare Earth Elements and Rare Metals Extractions

The stream that was taken out of 105 recirculation then was treated for rare earth elements and are metals extraction 108, in which the reduction of the remaining iron back to iron 2 (Fe²⁺), followed by a series of solvent extraction stages, was performed. The reactants were oxalic acid, NaOH, DEHPA (Di-(2-ethylhexyl)phosphoric acid) and TBP (tri-n-butyl phosphate) organic solution, kerosene, and HCl were used to convert rare earth element chlorides and rare metals chlorides to hydroxides. Countercurrent organic solvent with stripping of solution using HCl before proceeding to specific calcination from the rare earth elements and rare metals in form of hydroxide and conversion to high purity individual oxides. A ion exchange technique is also capable of achieving same results as polytrimethylene terephthalate (PET) membrane.

Iron powder from 105, or scrap metal as FeO, can be used at a rate dependent on Fe³⁺ concentration in the mother liquor.

34

HCl (100% wt) at the rate of 1 tph can be required as the stripped solution in REE Solvent Extraction (SX) separation and re-leaching of rare earth elements and/or rare metals oxalates.

5 Water of very high quality, demineralized or nano, at the rate of 100 tph was added to the strip solution and washing of precipitates.

Oxalic acid as di-hydrate at a rate of 0.2 tph was added and contributes to the rare earth elements and rare metals oxalates precipitation. NaOH or MgOH at a rate of 0.5 tph can be used as a neutralization agent.

DEHPA SX organic solution at the rate of 500 g/h was used as active reagent in rare earth elements separation while TBP SX organic solution at the rate of 5 kg/h is used as the active reagent for gallium recovery and yttrium separation. Finally, a kerosene diluent was used at the rate of approximately 2 kg/h in all SX section. Calcination occurs in an electric rotary furnace via indirect heating to convert contents to REE₂O₃ (oxides form) and maintain product purity.

Results of various tests made regarding stage 108 are shown in Table 6.

One line divided in subsections (5) to isolate the following elements using solvent extraction:

25 Ga₂O₃
Y₂O₃
Sc₂O₃
Eu₂O₃+Er₂O₃+Dy₂O₃
30 Ce₂O₃+Nd₂O₃+Pr₂O₃

Equivalent output earths oxides 166.14 kg/h Projected production as per pilot testing results		
Feed	Incoming (kg/h)	Final extraction individual (kg/h)
Ga ₂ O ₃	15.66	11.98
Sc ₂ O ₃	9.06	8.11
Y ₂ O ₃	22.56	20.22
La ₂ O ₃	32.24	25.67
Ce ₂ O ₃	61.37	51.82
Pr ₂ O ₃	8.08	6.18
Nd ₂ O ₃	30.3	27.24
Sm ₂ O ₃	5.7	4.51
Eu ₂ O ₃	1.06	0.95
Gd ₂ O ₃	4.5	4.06
Dy ₂ O ₃	3.9	3.55
Er ₂ O ₃	2.1	1.86
Total	196.55	166.14

50 Global Yield: 84.53%

Alternatively, stage 108 can be carried out as described in WO/2012/126092 and/or WO/2012/149642, that are hereby incorporated by reference in their entirety.

The solution after stages 108 and 109 contained mainly MgCl₂, NaCl, KCl, CaCl₂, FeCl₂/FeCl₃, and AlCl₃ (traces), and then undergoes the 111 stage. Na, K, Ca that follows the MgO can be extracted in stage 110 by crystallization in a specific order; Na first, followed by K, and then Ca. This technique can be employed for example in the Israeli Dead Sea salt processing plant to produce MgO and remove alkali from the raw material.

HCl Regeneration

Alkali (Na, K), once crystallized, was sent and processed in the alkali hydrochloric acid regeneration plant 110 for recovering highly concentrated hydrochloric acid (HCl). The process chosen for the conversion can generate value-added products

Various options are available to convert NaCl and KCl with intent of recovering HCl. One example can be to contact them with highly concentrated sulfuric acid (H_2SO_4), which generates sodium sulphate (Na_2SO_4) and potassium sulfate (K_2SO_4), respectively, and regenerates HCl at a concentration above 90% wt. Another example, is the use of a sodium and potassium chloride brine solution as the feed material to adapted small chlor-alkali electrolysis cells. In this latter case, common bases (NaOH and KOH) and bleach (NaOCl and KOCl) are produced. The electrolysis of both NaCl and KCl brine is done in different cells where the current is adjusted to meet the required chemical reaction. In both cases, it is a two-step process in which the brine is submitted to high current and base (NaOH or KOH) is produced with chlorine (Cl_2) and hydrogen (H_2). H_2 and Cl_2 are then submitted to a common flame where highly concentrated acid in gas (100% wt.) phase is produced and can be used directly in the crystallization stage **104**, or to crystallization stages requiring dry highly concentrated acid.

Magnesium Oxide

The reduced flow, which was substantially free of most elements (for example AlCl_3 , FeCl_3 , REE-Cl, NaCl, KCl) and rich in MgCl_2 , was then submitted to the magnesium oxides plant **111**. In the MgO , pyrohydrolysis of MgCl_2 and any other leftover impurities were converted into oxide while regenerating acid. The first step was a pre-evaporator/crystallizer stage in which calcium is removed and converted into gypsum ($\text{CaSO}_4 \cdot 2\text{H}_2\text{O}$) by a simple chemical reaction with sulfuric acid, for which separation of MgO is required. This increases the capacity of MgO roasting and also energy consumption slightly, while substantially recovering HCl. The next step was the specific pyrohydrolysis of MgO concentrated solution by spray roasting. Two (2) main products were generated; MgO that was further treated and HCl (about 18% wt.), which was either recycled back to the upstream leaching stage **102** or to the hydrochloric acid purification plant (**107**). The MgO -product derived from the spray roaster can require further washing, purification, and finally calcining depending on the quality targeted. The purification and calcining can comprise a washing-hydration step and standard calcining step.

The MgO from the spray roaster is highly chemically active and was directly charged into a water tank where it reacts with water to form magnesium hydroxide, which has poor solubility in water. The remaining traces of chlorides, like MgCl_2 , NaCl, dissolved in water. The $\text{Mg}(\text{OH})_2$ suspension, after settling in a thickener, was forwarded to vacuum drum filters, which remove the remaining water. The cleaned $\text{Mg}(\text{OH})_2$ is then forwarded into a calcination reactor where it is exposed to high temperatures in a vertical multi-stage furnace. Water from hydration is released and allows the transformation of the $\text{Mg}(\text{OH})_2$ to MgO and water. At this point, the magnesium oxide was of high purity (>99%).

HCl Purification

The hydrochloric acid purification stage **107** is effective for purifying HCl regenerated from different sectors (for example **105**, **106**, **111**) and to increase its purity for crystallization, whereas dry highly concentrated acid (>90% wt.) can be used as the sparging agent. Stage **107** also allowed for controlling the concentration of the acid going back to stage **102** (about 22 to about 32% wt.) and allows total acid and water balance. Total plant water balance is performed mainly by reusing wash water as absorption medium, as quench agent or as dissolution medium at the crystallization stages.

For example, HCl purification can be carried out as shown in FIGS. **4** and **5**.

For example, purification can be carried out by means of a membrane distillation process. The membrane distillation process applied here occurs when two aqueous liquids with different temperatures are separated through a hydrophobic membrane. The driving force of the process was supplied by the partial pressure vapour difference caused by the temperature gradient between these solutions. Vapour travels from the warm to the cold side. Without wishing to be bound to such a theory, the separation mechanism was based on the vapour/liquid equilibrium of the HCl/water liquid mixture. Practical application of such a technology has been applied to HCl/water, H_2SO_4 /water systems and also on large commercial scales on aqueous solution of sodium chloride with the purpose of obtaining potable water from seawater and nano water production. Therefore membrane distillation was a separation process based on evaporation through a porous hydrophobic membrane. The process was performed at about 60° C. and was effective to recover heat from the **104** and **102** stage with an internal water circulation loop, in order to maintain a constant incoming temperature to the membranes. For example, eight membranes of 300,000 m^2 equivalent surface area can be used per membrane to obtain a concentration of HCl well above the azeotropic point (i.e. >36%) of the $\approx 750 \text{ m}^3/\text{h}$ and final 90% concentration is then obtained through pressure distillation (rectification column).

Purification of HCl by processing thus regenerated acid through hydrophobic membrane and separating water from HCl; therefore increasing HCl concentration up to about 36% (above azeotropic point) and therefore allowing with a single stage of rectification through a pressure stripping column to obtain >90% in gaseous phase, for crystallization stage (sparging); and therefore controlling acid concentration into crystallization stages up to 30-35%_(aq).

As indicated stage **107** was operated at about 60° C. and heat input provided by heat recovery from stages **102** to **110**. Rectification column was operated at about 140° C. in the reboiler part. Net energy requirement was neutral (negative in fact at $-3.5 \text{ GJ/t Al}_2\text{O}_3$) since both systems were in equilibrium and in balance.

For example, the acid purification can be carried out by using adsorption technology over an activated alumina bed. In continuous mode, at least two adsorption columns are required to achieve either adsorption in one of them and regeneration in the other one. Regeneration can be performed by feeding in counter-current a hot or depressurized gas. This technology will result in a purified gas at 100% wt.

For example, the acid purification can be made by using calcium chloride as entrainer of water. A lean hydrochloric acid solution is contacted with a strong calcium chloride solution through a column. The water is then removed from the hydrochloric acid solution and 99.9% gaseous HCl comes out of the process. Cooling water and cryogenic coolant is used to condense water traces in the HCl. The weak CaCl_2 solution is concentrated by an evaporator that ensures the recuperation of calcium chloride. Depending on the impurities in the incoming HCl solution feed to the column, some metals can contaminate the calcium chloride concentrated solution. A precipitation with $\text{Ca}(\text{OH})_2$ and a filtration allows the removal of those impurities. The column can operate for example at 0.5 barg. This technology can allow for the recuperation of 98% of the HCl.

Table 7 shows the results obtained concerning the process shown in FIG. **6**.

Composition	Stage 101	Stage 102	Stage 106	Stage 105	MgO		Stage 107	Stage 108	TOTAL PRODUCED
(% wt)	Yield (%)	Yield (%)	Yield (%)	Yield (%)	tpy	Yield (%)	Yield (%)	Yield (%)	Yield (%)
Main constituents									
SiO ₂	—	99.997%	—	—	—	—	—	—	99.997%
Al	—	98.02%	95.03%	—	—	—	—	—	95.03%
Fe	—	100.00%	—	92.65%	—	—	—	—	92.65%
Mg	—	99.998%	—	—	29,756	92.64%	—	—	92.64%
Ca	—	99.998%	—	—	—	—	—	—	98.28%
Na	—	99.998%	—	—	—	—	—	—	92.76%
K	—	100.00%	—	—	—	—	—	—	93.97%
Others incl. H ₂ O	—	—	—	—	—	—	—	—	—
RE/RM	—	99.80%	—	92.32%	—	—	—	84.67%	84.67%
By-Products									
NaOH	—	—	—	—	68,556	—	—	—	—
NaOCl	—	—	—	—	9,269	—	—	—	—
KOH	—	—	—	—	73,211	—	—	—	—
KOCl	—	—	—	—	9,586	—	—	—	—
CaSO ₄	—	—	—	—	46,837	—	—	—	—
Reactants									
H ₂ SO ₄ (*)	—	—	—	—	19,204	—	—	—	—
Fresh HCl M-UP	—	—	—	—	—	—	99.75%	—	99.75%
Total	—	98.55%	95.03%	—	256,419	92.64%	99.75%	84.67%	—

Tables 8 to 26 show results obtained concerning the products made in accordance with the process shown in FIG. 6 in comparison with standard of the industry.

TABLE 8

Chemical composition of obtained alumina		
Element	% Weight*	Standard used in industry
Al ₂ O ₃	99.938	98.35 min
Fe ₂ O ₃	0.0033	0.0100
SiO ₂	0.0032	0.0150
TiO ₂	0.0003	0.0030
V ₂ O ₅	0.0008	0.0020
ZnO	0.0005	0.0030
Cr ₂ O ₃	0.0003	N/A
MgO	0.0090	N/A
MnO	0.0002	N/A
P ₂ O ₅	0.0039	0.0010
Cu	0.0030	N/A
Ca	0.0020	0.0030
Na	0.0190	0.4000
K	0.0053	0.0150
Li	0.0009	N/A
Ba	<0.00001	0.0000
Th	<0.000001	0.0000
U	<0.000001	0.0000
Free Cl ⁻	Not detectable	0.0000
LOI	<1.0000	<1.0000

P₂O₅ removal technique can include, for example, after leaching, phosphorous precipitation using zirconium sulphate. It can be provided, for example, in a solution heated at 80 to about 90° C. or about 85 to about 95° C., under vacuum.

TABLE 9

Physical properties of obtained alumina		
Property	Orbite Alumina	Standard used in industry
PSD < 20 μm	5-10%	N/A
PSD < 45 μm	10-12%	<10%

TABLE 9-continued

Physical properties of obtained alumina		
Property	Orbite Alumina	Standard used in industry
PSD > 75 μm	50-60%	N/A
SSA (m ² /g)	60-85	60-80
Att. Index	10-12%	<10%
α Al ₂ O ₃	2-5%	<7-9%

TABLE 10

Chemical composition of obtained hematite		
Element	% Weight	
Fe ₂ O ₃	>99.5%	
Hydrolysable elements	<0.2%	

TABLE 11

Physical properties of obtained hematite*	
Property	Orbite hematite
PSD _{mean}	25-35 μm
Density (bulk)	2000-3000 kg/m ³
Humidity after filtration	<10%

*Material can be produced as bricks

TABLE 12

Chemical composition of obtained silica	
Element	% Weight
SiO ₂	>99.7
Al ₂ O ₃	<0.25%
MgO	≈0.1%
Fe ₂ O ₃	≈0.1%

39

TABLE 12-continued

Chemical composition of obtained silica	
Element	% Weight
CaO	≈0.01%
Na ₂ O	<0.1%
K ₂ O	<0.1%

Note:

Product may have unbleached cellulose fiber filter aid. Cellulose wood flour.

TABLE 13

Physical properties of obtained silica	
Property	Orbite silica
PSD _{mean}	10-20 μm
Specific surface area	34 m ² /g
Density (bulk)	2000-2500 kg/m ³
Humidity after filtration	<30%

TABLE 14

Purity of obtained rare earth element oxides	
Element	Purity (%)
Ga ₂ O ₃	>99%
Sc ₂ O ₃	
Y ₂ O ₃	
La ₂ O ₃	
Ce ₂ O ₃	
Pr ₂ O ₃	
Nd ₂ O ₃	
Sm ₂ O ₃	
Eu ₂ O ₃	
Gd ₂ O ₃	
Dy ₂ O ₃	
Er ₂ O ₃	
Physical properties of obtained REE-O/RM-O	
Property	Orbite REE-O/RM-O
PSD _{mean}	2-30 μm
Density	5500-13000 kg/m ³
LOI	<1%

TABLE 15

Chemical composition of obtained MgO		
Element	Typical	Specification
MgO	99.0+	98.35 min
CaO	0.0020	0.83
SiO ₂	0.0000	0.20 max
B ₂ O ₃	0.0000	0.02 max
Al ₂ O ₃	0.0300	0.12 max
Fe ₂ O ₃	0.0160	0.57 max
MnO ₂	<0.14	0.14 max
LOI	0.7%	<1%

40

TABLE 16

Physical properties of obtained MgO	
Property	Orbite MgO
PSD _{mean}	10 μm
Density	N/A
LOI	650 kg/m ³

TABLE 17

Chemical composition of obtained NaOH	
Element	% Weight
Sodium hydroxide	32%
Water	68%

TABLE 18

Physical properties of obtained NaOH	
Property	Sodium hydroxide (NaOH)
Physical state	Liquid
Vapour pressure	14 mmHg
Viscosity	>1
Boiling point	100° C.
Melting point	0° C.
Specific gravity	1.0

TABLE 19

Chemical composition of obtained sodium hypochlorite (bleach)	
Element	% Weight
Sodium hypochlorite	12%
Sodium hydroxide	<1%
Water	>80%

TABLE 20

Physical properties of obtained NaOCl	
Property	Sodium hypochlorite (NaOCl)
Physical state	Liquid
Vapour pressure	1.6 kPa
Viscosity	N/A
Boiling point	100° C.
Melting point	-3° C.
Specific gravity	1.2

TABLE 21

Chemical composition of obtained potassium hydroxide	
Element	% Weight
Potassium hydroxide	32%
Water	68%

41

TABLE 22

Physical properties of obtained potassium hydroxide	
Property	KOH
Physical state	Liquid
Vapour pressure	17.5 mmHg
Viscosity	N/A
Boiling point	100° C.
Melting point	N/A
Specific gravity	1.18

TABLE 23

Chemical composition of obtained potassium hypochlorite (KOCl)	
Element	% Weight
Potassium hypochlorite	12%
Potassium hydroxide	<1%
Water	>80%

TABLE 24

Physical properties of obtained potassium hypochlorite	
Property	KOCl
Physical state	Liquid
Vapour pressure	N/A
Viscosity	N/A
Boiling point	103° C.
Melting point	N/A
Specific gravity	>1.0

TABLE 25

Chemical composition of obtained calcium sulphate dihydrate	
Element	% Weight
Calcium sulphate dehydrate	100%

TABLE 26

Physical properties of obtained calcium sulphate dehydrate	
Property	Orbite CaSO ₄ •2H ₂ O
Physical state	Solid
Specific gravity	2.32

In order to demonstrate the versatility of the processes of the present disclosure, several other tests have been made so as to shown that these processes can be applied to various sources of starting material.

Example 6

Another starting material has been used for preparing acidic compositions comprising various components. In fact, a material that is a concentrate of rare earth elements and rare metals (particularly rich in zirconium) has been tested. Table 27 shows the results carried out on such a starting material using a similar process as shown in FIGS. 1, 3, 6 and 7 and as detailed in Examples 1, 2 and 5. It can thus be inferred from the results shown in Table 27 that the various components present in the leaching (various metals such as aluminum,

42

iron, magnesium as well as rare earth elements and rare metals) can be extracted from the obtained leaching composition and that they can eventually be isolated by the processes of the present disclosure such as, for example, those presented in Examples 1, 2 and 5.

Example 7

Other tests have been made in a similar manner as described in Example 6. In the present example, carbonatite has been used as a starting material. (see Table 28 below).

TABLE 27

Tests made on a zirconium rich material.					
Raw material	Composition measure and/or evaluated (% wt.)	Average measured for testing (% wt.)	Extraction rate measured (ALP) (%)	O All process recovery (%)	Orbite recovery (%)
Al ₂ O ₃	6.12	6.12	89.65	86.95	86.95
Fe ₂ O ₃	15.80	15.80	99.50	97.51	97.51
SiO ₂	36.00	36.00	0.000	99.997	99.997
MgO	3.08	3.08	99.75	92.66	92.66
Na ₂ O	1.13	1.13	99.50	99.50	99.50
K ₂ O	2.12	2.12	99.50	99.50	99.50
CaO	6.10	6.10	99.50	99.00	99.00
S total	0.22	0.22		100.00	100.00
F	1.98	1.98	99.50	99.00	99.00
TiO ₂	0.13	0.13	0.000	99.03	99.03
V ₂ O ₅	0.00	0.00	98.00	96.04	96.04
P ₂ O ₅	1.10	1.10	98.00	96.04	96.04
MnO	0.43	0.43	98.00	96.04	96.04
ZrO ₂	12.43	12.43	22.70	20.43	20.43
Cr ₂ O ₃	0.00	0.00	0.00	0.00	0.00
Ce ₂ O ₃	3.05	3.045	97.31	92.98	92.98
La ₂ O ₃	1.34	1.337	99.55	92.68	92.68
Nd ₂ O ₃	1.55	1.551	98.40	94.79	94.79
Pr ₂ O ₃	0.37	0.375	99.75	97.52	97.52
Sm ₂ O ₃	0.15	0.151	88.75	84.80	84.80
Dy ₂ O ₃	0.09	0.089	80.35	76.77	76.77
Er ₂ O ₃	0.03	0.030	72.60	69.37	69.37
Eu ₂ O ₃	0.03	0.027	85.57	81.76	81.76
Gd ₂ O ₃	0.21	0.205	82.85	79.16	79.16
Ho ₂ O ₃	0.01	0.013	77.10	73.67	73.67
Lu ₂ O ₃	0.00	0.003	60.15	57.47	57.47
Tb ₂ O ₃	0.02	0.022	78.05	74.58	74.58
Th	0.02	0.022	88.10	84.18	84.18
Tm ₂ O ₃	0.00	0.004	66.85	63.88	63.88
U	0.01	0.014	81.90	78.26	78.26
Y ₂ O ₃	0.30	0.300	72.70	69.46	69.46
Yb ₂ O ₃	0.02	0.023	62.80	60.01	60.01
Ga ₂ O ₃	0.02	0.016	96.90	92.59	92.59
Sc ₂ O ₃	0.00	0.003	95.00	90.77	90.77
LOI (inc. water)	6.122023973	6.12			

TABLE 28

Tests made on carbonatite					
Raw material	Composition measure and/or evaluated (% wt.)	Average measured for testing (% wt.)	Extraction rate measured (ALP) (%)	O All process recovery (%)	Orbite recovery (%)
Al ₂ O ₃	0.70	0.70	84.31	81.61	81.61
Fe ₂ O ₃	11.22	11.22	94.14	92.15	92.15
SiO ₂	2.11	2.11	0.00003	99.997	99.997
MgO	6.50	6.500	100	96.25	96.25
Na ₂ O	0.07	0.07	92.54	90.55	90.55
K ₂ O	0.18	0.181	37.33	37.33	37.33
CaO	16.51	16.51	100	98.00	98.00
TiO ₂	0.00	0.000	0.00000	100.000	100.000
V ₂ O ₅	0.00	0.000	0	100.000	100.000

TABLE 28-continued

Tests made on carbonatite				
Raw material	Composition measure and/or evaluated (% wt.)	Average measured for testing (% wt.)	Extraction rate measured (ALP) (%)	O All Orbit process recovery (%)
P ₂ O ₅	0.00	0.000	0	100.000
MnO	0.00	0.000	0	100.000
ZrO ₂	0.00	0.000	0	100.000
Cr ₂ O ₃	0.00	0.000	0	100.000
Ce ₂ O ₃	1.19	1.195	64.04	61.190
La ₂ O ₃	0.46	0.463	63.86	61.018
Nd ₂ O ₃	0.45	0.448	81.46	77.835
Pr ₂ O ₃	0.14	0.142	67.59	64.582
Sm ₂ O ₃	0.03	0.033	65.32	62.413
Dy ₂ O ₃	0.00	0.000	78.12	74.644
Er ₂ O ₃	0.00	0.000	86.15	82.316
Eu ₂ O ₃	0.01	0.007	66.45	63.493
Gd ₂ O ₃	0.01	0.013	54.46	52.037
Ho ₂ O ₃	0.00	0.000	83.12	79.421
Lu ₂ O ₃	0.00	0.000	88.86	84.906
Tb ₂ O ₃	0.00	0.001	41.42	39.577
Th	0.06	0.065		
Tm ₂ O ₃	0.00	0.000	90.70	86.664
U	0.01	0.007		
Y ₂ O ₃	0.00	0.000	84.68	80.912
Yb ₂ O ₃	0.00	0.000	85.11	81.323
Ga ₂ O ₃	0.00	0.000	0	0.000
Sc ₂ O ₃	0.00	0.000	0	0.000
LOI (inc. water)		60.33		

It can thus be inferred from the results shown in Table 28 that the various metals, rare earth elements and rare metals extracted present in the obtained leaching composition can eventually be isolated by the processes of the present disclosure such as, for example, those presented in Examples 1, 2 and 5.

The process shown in FIG. 8 is similar to the process of FIG. 1, with the exception that in FIG. 8, the term “aluminum” is replaced by a “first metal”. The person skilled in the art would thus understand that in accordance with the present disclosure, the processes can also encompass recovering various other products and using various types of material as starting material. The first metal can be chosen from Al, Fe, Ti, Zn, Ni, Co, Mg, Li, Mn, Cu, Au, Ag, Pd, Pt. and mixtures thereof etc. Such a process can thus be used for recovering various other metals than aluminum. Thus, the first metal will be precipitated as a chloride in stage 5 and eventually converted into an oxide.

In fact, the person skilled in the art would understand that by replacing in FIGS. 1, 3, 6 and 7 the term “aluminum” with the expression “first metal” the processes shown in these figures can be used to obtain various other products than alumina and also used for treating various different starting material. Thus, the first metal can be recovered as a chloride (as it is the case for aluminum chlorides in the processes of FIGS. 1, 3, 6 and 7) and all the other stages of these processes can thus be carried out (when applicable) depending on the nature of the starting material used.

In step 4, the first metal chloride can be precipitated or crystallized. In fact, the first metal can be removed from the leachate in various manner. For example, a precipitating agent can be added or HCl (for example gaseous) can be reacted with the liquid obtained from step 3 so as to cause precipitation and/or crystallization of the first metal chloride. Alternatively, the temperature of the leachate can be controlled so as to substantially selectively cause precipitation of the first metal chloride. This can also be done, for example, by using an ion exchange resin, by adding a precipitating agent,

etc. Combinations of various possibilities mentioned in this paragraph can also be envisaged. All these methods of at least substantially selectively recovering the first metal chloride can also be applied on the first metal oxide. They can be applied to only the first metal chloride, only to the second metal oxide or to both of them. In fact, the liquid obtained after stage 4 can be treated so as to convert the second metal chloride into the corresponding oxide and then, the second metal oxide can be substantially selectively recovered by selectively precipitating and/or crystallizing such a second metal oxide, for example by controlling the temperature, by using an ion exchange resin, by adding a precipitating agent, by adding HCl, or mixtures thereof.

As previously indicated, the processes of the present disclosure can be efficient for treating material comprising Al, Fe, Ti, Zn, Ni, Co, Mg, Li, Mn, Cu, Au, Ag, Pd, Pt.

For example, when treating a material that comprises, for example, Mg and Fe, the material can be leached for example by using HCl. Then, while the mixture (comprising a solid and a liquid) so obtained is still hot, it can be treated so as to separate the solid from the liquid (for example by means of a solid/liquid separation). That will be effective for removing solids such as Si and optionally others such as Ti. Thus, the liquid can be cooled down to a temperature of about 5 to about 70° C., about 10 to about 60° C., about 10 to about 50° C., about 10 to about 40° C., or about 15 to about 30° C. so as to substantially selectively precipitate or crystallize magnesium (for example as MgCl₂ (first metal chloride in FIG. 8)), as shown in 4 of such a figure. Then, the first metal chloride can be converted as shown in 5 so as to obtain the first metal oxide. The iron can then be treated as in 6 of FIG. 8. The remainder of the process shown in FIG. 8 (stages 6 to 10) being as described previously for FIG. 1.

In a similar manner, the expression “iron oxides” in step 6 can be replaced in FIG. 8 by “second metal oxide”. The person skilled in the art would thus understand that another metal than iron could be recovered as an oxide pursuant, for example, to a treatment of the liquid obtained from step 4. As previously discussed, the chloride of the second metal obtained from step 4 can be treated so as to substantially selectively recover the chloride of the second metal. This can be done, for example, by selectively precipitating and/or crystallizing such a second metal chloride by controlling the temperature, by substantially selectively hydrolyzing such a second metal chloride, by using an ion exchange resin, by adding a precipitating agent, by adding HCl, or mixtures thereof. All these methods of at least substantially selectively recovering the second metal chloride can also be applied on the second metal oxide. The can be applied to only the second metal chloride, only to the second metal oxide or to both of them. In fact, the liquid obtained after stage 4 can be treated so as to convert the second metal chloride into the corresponding oxide and then, the second metal oxide can be substantially selectively recovered by selectively precipitating and/or crystallizing such a second metal oxide, for example by controlling the temperature, by using an ion exchange resin, by adding a precipitating agent, by adding HCl, or mixtures thereof.

Alternatively, the iron chloride can be treated by means of a pyrohydrolysis in order to generate Cl₂. In such a case, the Cl₂ so produced can be used in step 13 of FIG. 8 for treating the solid that comprises Ti. For example, it can be used to convert TiO₂ into TiCl₄. Such a reaction can be carried under oxidizing conditions (for example more than 4% of O₂). The pyrohydrolysis can be carried out in a fluid bed reactor under such oxidizing conditions.

45

For example, platinum and palladium can also be treated similarly. Moreover, their separation can also be accomplished with ion exchange: selective crystallization in HCl is possible and can be temperature sensitive.

FIGS. 9A and 9B show methods for separating Si from Ti. For example, when using an ore as starting material, leaching can be carried out in the presence of Cl_2 (optionally in the presence of carbon) so as to maintain Ti under the form of TiCl_4 since it remains in solution (fluid) while Si remains solid (SiO_2). Then, Ti (such as TiCl_4) can be heated so as to be converted into TiO_2 . For example, it can be injected into a plasma torch for being purified.

Such a method for purifying Si and Ti can be used in all the processes of the present disclosure when there is a need for separating these two entities. See stage 13 in FIGS. 1, 3 and 8 and stage 113 in FIGS. 6 and 7.

The processes of the present disclosure provide a plurality of important advantages and distinction over the known processes. This is true for the production of various products has previously indicated.

The processes of the present disclosure can provide economical solutions that can successfully extract various products from various type of materials while providing ultra pure secondary products of high added value including highly concentrated rare earth elements and rare metals. The technology described in the present disclosure can allow for an innovative amount of total acid recovery and also for a ultra high concentration of recovered acid. When combining it to the fact that combined with a semi-continuous leaching approach that favors very high extraction yields and allows a specific method of crystallization of some products, and concentration of other value added elements.

For example, the crystallization of AlCl_3 into $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$ using dried, cleaned and highly concentrated gaseous HCl as the sparging agent allows for a pure aluminum chloride hexahydrate with only few parts per million of iron and other impurities. A minimal number of stages are required to allow proper crystal growth.

The direct interconnection with the calcination of $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$ into Al_2O_3 which does produce very high concentration of gas allows the exact adjustment in continuous of the HCl concentration within the crystallizer and thus proper control of the crystal growth and crystallization process.

The applicants have now discovered fully integrated and continuous processes with substantially total hydrochloric acid recovery for the extraction of alumina and other value added products from various materials that contain aluminum (clay, bauxite, aluminosilicate materials, slag, red mud, fly ashes etc.) containing aluminum. In fact, the processes allows for the production of substantially pure alumina and other value added products purified such as purified silica, pure hematite, titanium oxide, pure other minerals (ex: magnesium oxide) and rare earth elements products. In addition, the processes do not necessary require thermal pre-treatment before the acid leach operation. Acid leach can be carried out using semi-continuous techniques with high pressure and temperature conditions and very high regenerated hydrochloric acid concentration. In addition, the processes do not generate any residues not sellable, thus eliminating harmful residues to environment like in the case of alkaline processes.

The advantage of the high temperature calcination stage, in addition for allowing to control the α -form of alumina required, can be effective for providing a concentration of hydrochloric acid in the aqueous form (>38%) that is higher than the concentration of HCl at the azeotropic point and thus providing a higher incoming HCl concentration to the leaching stage. The calcination stage hydrochloric acid network

46

can be interconnected to two (2) crystallization systems and by pressure regulation excess HCl can be being absorbed at the highest possible aqueous concentration. The advantage of having a hexahydrate chloride with low moisture content (<2%) incoming feed can allow for a continuous basis to recover acid at a concentration that is higher than the azeotropic concentration. This HCl balance and double usage into three (3) common parts of the processes and above azeotropic point is a substantial advance in the art.

Another advantage can be the use of the incoming chemistry (ferric chloride) to the iron oxide and hydrochloric acid recovery unit where all excess heat load from any calcination part, pyrohydrolysis and leaching part is being recovered to pre-concentrate the mother liquor in metal chloride, thus allowing, at very low temperature, the hydrolysis of the ferric chloride in the form of very pure hematite and the acid regeneration at the same concentration than at its azeotropic point.

For example, the processes of the present disclosure can be effective for providing, at the ferric chloride hydrolysis step, the possibility to concentrate rare earth elements in form of chlorides at very high concentration within the hydrolyser reactor through an internal loop between hydrolyzer and crystallization. The advantage in that the processes of the present disclosure benefit from the various steps where gradual concentration ratios are applied. Thus, at this stage, in addition to an internal concentration loop, having the silica, the aluminum, the iron and having in equilibrium a solution close to saturation (large amount of water evaporated, no presence of free hydrochloric acid) allows for taking rare earth elements and non-hydrolysable elements in parts per million into the incoming feed and to concentrate them in high percentage directly at the hydrolyser after ferric chloride removal. Purification of the specific oxides (RE-O) can then be performed using various techniques when in percentage levels. The advantage is doubled here: concentration at very high level of rare earth elements using integrated process stages and most importantly the approach prevents from having the main stream (very diluted) of spent acid after the leaching step with the risk of contaminating the main aluminum chloride stream and thus affecting yields in Al_2O_3 . Another important improvement of the art is that on top of being fully integrated, selective removal of components allows for the concentration of rare earth elements to relatively high concentration (percentages).

Another advantage of the processes can be a selective crystallization of MgCl_2 through the sparging of HCl. Highly concentrated acid both in gaseous phase or in aqueous form can thus be generated. As previously indicated, $\text{Mg}(\text{OH})_2$ can also be obtained. As per aluminum chloride specific crystallization, the direct interconnection with the calcination reactor, the HCl gas very high concentration (about 85 to about 95%, about 90 to 95% or about 90% by weight) can allow for exact adjustment in continuous of the crystallizer based on quality of magnesium oxide targeted.

The pyrohydrolysis can allow for conversion of any remaining chloride and the production of refined oxides that can be used (in case of clay as starting material) as a fertilizer and allowing the processing of large amount of wash water from the processes with the recovery hydrochloric acid in close loop at the azeotropic point for the leaching step. This is related to the fact that it does totally close the process loop in terms of acid recovery and the insurance that no residues harmful to the environment are being generated while processing any type of raw material, as previously described.

It was demonstrated that the processes of the present disclosure offer effective solutions for extracting various metals from various materials. For example, it was demonstrated that

materials that contain titanium can be efficiently treated so as to recover titanium therefrom while also recovering other metals as those described in the present disclosure.

It was thus demonstrated that the present disclosure provides fully integrated processes for the extraction and/or recovery of titanium and/or for the preparation of titanium chloride or titanium oxide using a hydrochloric acid treatment while producing high purity and high quality products (minerals) and optionally extracting rare earth elements and rare metals.

With respect to the above-mentioned examples 1 to 5, the person skilled in the art will also understand that depending on the starting material used some parameters might need to be adjusted consequently. In fact, for example, certain parameters such as reaction time, concentration, temperature may vary in accordance with the reactivity of the selected starting material.

While a description was made with particular reference to the specific embodiments, it will be understood that numerous modifications thereto will appear to those skilled in the art. Accordingly, the above description and accompanying drawings should be taken as specific examples and not in a limiting sense.

What is claimed is:

1. A process for preparing various products, said process comprising:
 - leaching a raw aluminum-containing material with HCl so as to obtain a leachate comprising Al, Fe and Mg ions and a solid comprising Si, and separating said solid from said leachate, thereby recovering Si;
 - recovering said Al from said leachate by recovering Al as precipitated AlCl_3 , by controlling the temperature of said leachate and/or concentration of HCl, by using a precipitating agent, or by using an ion exchange resin;
 - recovering said Fe from said leachate by recovering Fe as precipitated Fe_2O_3 , by controlling the temperature of said leachate and/or concentration of HCl, by carrying out an hydrolysis, by using a precipitating agent, or by using an ion exchange resin; and
 - recovering said Mg from said leachate by recovering Mg as precipitated MgCl_2 , by controlling the temperature of said leachate and/or concentration of HCl, by using a precipitating agent, or by using an ion exchange resin.
2. The process of claim 1, wherein said aluminum-containing material is leached with HCl having a concentration of about 25 to about 45 weight %.
3. The process of claim 1, wherein said aluminum-containing material is leached with HCl having a concentration of about 25 to about 45 weight % at a temperature of about 125 to about 225° C.
4. The process of claim 1, wherein said process further comprises calcining at least one of AlCl_3 and MgCl_2 and recycling gaseous HCl so-produced by contacting it with water so as to obtain a composition having a concentration of about 25 to about 45 weight % and using said composition for leaching said aluminum-containing material.
5. The process of claim 4, wherein said recycled gaseous HCl so-produced is contacted with water so as to obtain said composition having a concentration of about 26 to about 42 weight % and said composition is reacted, at a temperature of about 125 to about 225° C., with said aluminum-containing material so as to leaching it.
6. The process of claim 1, wherein said leachate comprises iron chloride and is concentrated to a concentrated liquid having an iron chloride concentration of at least 30% by weight; and then hydrolyzed at a temperature of about 155 to about 350° C.

7. The process of claim 1, further comprising calcining MgCl_2 into MgO.

8. The process of claim 1, wherein recovering said Al comprises reacting said leachate with gaseous HCl so as to obtain said liquid and said precipitate comprising said aluminum ions in the form of $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$.

9. The process of claim 8, wherein said process comprises converting $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$ into Al_2O_3 by carrying out a calcination of $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$.

10. The process of claim 8, wherein said process comprises converting $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$ into Al_2O_3 by heating said $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$ at a temperature of at least 1200° C.

11. The process of claim 8, wherein said process comprises converting $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$ into Al_2O_3 by heating said $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$ at a temperature of at least 350° C.

12. The process of claim 9, wherein converting $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$ into Al_2O_3 is carried out in two stages.

13. The process of claim 9, wherein converting $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$ into Al_2O_3 is carried out by injecting steam.

14. The process of claim 1, wherein said process comprises:

leaching said aluminum-containing material with HCl so as to obtain said leachate comprising Al, Fe and Mg ions and a solid comprising Si, and separating said solid from said leachate, thereby recovering Si;

recovering said Al from said leachate in the form of precipitated AlCl_3 by controlling the temperature of said leachate and/or concentration of HCl;

recovering said Fe from said leachate in the form of precipitated Fe_2O_3 by controlling the temperature of said leachate and/or concentration of HCl; and

recovering said Mg from said leachate by using gaseous HCl for crystallizing MgCl_2 .

15. A process for preparing aluminum, said process comprising:

obtaining AlCl_3 by a process as defined in claim 1;

converting said AlCl_3 into Al_2O_3 ; and

converting said Al_2O_3 into aluminum.

16. A process for preparing various products, the process comprising:

leaching a titanium-containing material comprising Ti, Si, Fe and Al with HCl so as to obtain a leachate comprising ions of Fe and Al and a solid comprising Ti and Si, and separating the solid from the leachate;

reacting the leachate with HCl so as to obtain a liquid comprising at least one chloride of Fe and a precipitate comprising a chloride of Al, and separating the precipitate from the liquid;

treating the liquid under conditions effective for converting the at least one chloride of Fe into Fe_2O_3 and recovering said Fe_2O_3 ;

heating the precipitate under conditions effective for converting the precipitate into Al_2O_3 and recovering said Al_2O_3 ;

treating said solid so as to substantially separate Si from Ti that are contained therein by:

leaching said solid with an acid in the presence of a chloride that was combined with said solid and said acid so as to obtain a liquid portion comprising Ti and a solid portion containing Si and wherein the liquid portion is separated from the solid portion so as to recover said Ti and said Si; or

treating said solid with Cl_2 in order to obtain a liquid portion comprising said Ti and a solid portion comprising said Si, and wherein said solid portion and said liquid portion are separated from one another so as to recover said Ti and said Si,

thereby separately recovering said Ti, Si, Fe and Al.

49

17. A process for preparing various products, the process comprising:

leaching a titanium-containing material comprising Ti, Al, Fe and Mg with HCl so as to obtain a leachate comprising ions of Al, Fe and Mg and a solid comprising Ti, and separating the solid from the leachate;

substantially selectively recovering said Al from said leachate;

substantially selectively recovering said Fe from said leachate;

substantially selectively recovering said Mg from said leachate;

treating said solid so as at least partially recover Ti.

18. The process of claim 17, wherein said process comprises controlling the temperature of said leachate and/or concentration of HCl so as to substantially selectively precipitate said Fe in the form of at least one chloride, and recovering said precipitate from said leachate.

19. The process of claim 17, wherein said process comprises:

controlling the temperature of said leachate and/or concentration of HCl so as to substantially selectively precipitate said Fe in the form of at least one chloride, and recovering said at least one chloride from said leachate;

reacting said leachate with gaseous HCl so as to substantially selectively precipitate said Al in the form of $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$, and recovering said $\text{AlCl}_3 \cdot 6\text{H}_2\text{O}$ from said leachate; and

reacting said leachate with gaseous HCl so as to substantially selectively precipitate said Mg in the form of MgCl_2 , and recovering said MgCl_2 from said leachate.

20. A process for preparing various products, said process comprising:

leaching an aluminum-containing material with HCl so as to obtain a leachate comprising Al, Fe, Ni and Mg ions and a solid comprising Si, and separating said solid from said leachate, thereby recovering Si;

recovering said Al contained in said leachate in the form of precipitated AlCl_3 by controlling the temperature of said leachate and/or concentration of HCl;

recovering said Fe contained in said leachate in the form of precipitated Fe_2O_3 by controlling the temperature of said leachate and/or concentration of HCl;

recovering said Mg contained in said leachate by crystallizing MgCl_2 ; and

50

recovering said Ni contained in said leachate under the form of NiCl_2 .

21. A process for preparing various products, said process comprising:

leaching an aluminum-containing material with HCl so as to obtain a leachate comprising Al, Fe and Mg ions and a solid comprising Si, and separating said solid from said leachate, thereby recovering Si;

recovering said Al from said leachate by recovering Al as precipitated AlCl_3 , by controlling the temperature of said leachate and/or concentration of HCl, by using a precipitating agent, or by using an ion exchange resin;

recovering said Fe from said leachate by recovering Fe as precipitated Fe_2O_3 , by controlling the concentration of HCl, by using a precipitating agent, or by using an ion exchange resin; and

recovering said Mg from said leachate by recovering Mg as precipitated MgCl_2 , by controlling the temperature of said leachate and/or concentration of HCl, by using a precipitating agent, or by using an ion exchange resin.

22. A process for preparing various products, said process comprising:

leaching an aluminum-containing material with HCl so as to obtain a leachate comprising Al, Fe and Mg ions and a solid comprising Si, and separating said solid from said leachate, thereby recovering Si;

substantially selectively recovering said Al from said leachate by recovering Al as precipitated AlCl_3 , by controlling the temperature of said leachate and/or concentration of HCl, by using a precipitating agent, or by using an ion exchange resin;

substantially selectively recovering said Fe from said leachate by recovering Fe as precipitated Fe_2O_3 , by controlling the temperature of said leachate and/or concentration of HCl, by carrying out an hydrolysis, by using a precipitating agent, or by using an ion exchange resin; and

substantially selectively recovering said Mg from said leachate by recovering Mg as precipitated MgCl_2 , by controlling the temperature of said leachate and/or concentration of HCl, by using a precipitating agent, or by using an ion exchange resin.

* * * * *